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THE MINERAL CHARACTERISTICS THAT AFFECT METAL RECOVERIES FROM THE Cu, Zn, Pb, AND Ag ORES FROM MANITOBA. PART 3.

A MINERALOGICAL EVALUATION OF THE BEHAVIOUR OF METALLIC MINERALS IN THE HBMS CONCENTRATOR CIRCUIT

R.E. Healy
University of Manitoba

W. Petruk
Mineral Processing Laboratory

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RAYMOND E. HEALY
Dept. of Geological Sciences, University of Manitoba,
Winnipeg, Manitoba, R3T 2N2.

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SCIENTIFIC AUTHORITY : Dr. W. Petruk, CANMET, EMR.

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ABSTRACT

Low grade Cu, Zn, Au and Ag ore fed to the HBMS concentrator circuit was supplied by the Centennial and South Main (Flin Flon) mines. Circuit samples were assayed for 15 elements, and image analysis was done to determine the modal quantities, grain size distributions and liberations of the minerals. Material balances were done to simulate the behaviour, and to determine the recovery of metals, minerals, and of free and unliberated grains of the valuable minerals at each point in the circuit.

Covellite and chalcocite are significant potential activators of sphalerite. These phases are not ore minerals, but are derived from the Cu-Fe-sulfides which comprise quenched Cu-matte. The latter is entrained in backfilled Cu-smelter slag, which is introduced into the feed ore by mine dilution. The slag silicate phases fayalite and glass together contain approximately 1.5 % Cu and 6.5 % Zn, masking Cu and Zn assays, the conventional method of evaluating the metallurgical performance of chalcopyrite and sphalerite, respectively. The slag, which comprised 9.4 Wt % of the feed, accounts for approximately 10 % of the Cu and 35 to 40 % of the Zn in the feed, and approximately 60 % of the Cu and 85 % of the Zn in the tailings. Thus, while the actual head grades are 1.04 % Cu and 1.80 % Zn, the 'effective' Cu and Zn head grades due to chalcopyrite and sphalerite, respectively are only approximately 0.9 % Cu and 1.2 % Zn.

Because the slag reports to the tailings, the recovery of Cu in the Cu-concentrate is only 77.5 %, whereas that of chalcopyrite is 89.2 %. Similarly, the recovery of Zn in the Zn-concentrate is only 40.2 %, whereas that of sphalerite is 71.0 %. Of the chalcopyrite in the feed 76.5 % occurs as free and sub-free grains, of which approximately 98.5 % report to the Cu-concentrate. Whereas 92.2 % of the middling grains are recovered in the Cu-concentrate, only 34.1 % of the sub-middling grains are recovered, with 61 % lost to the tailings. Thus, approximately 85 % of the chalcopyrite lost to the tailings occurs as sub-middling grains, and corresponds to approximately 8 % of the chalcopyrite in the feed. These losses are only recoverable through extremely fine regrinding of that fraction.

Of the sphalerite in the feed 69.4 % occurs as free and sub-free grains, of which approximately 81 % report to the Zn-concentrate, and 19 % to the tailings and Cu-concentrate. The loss of approximately 17 % of the free and sub-free grains to the Cu-concentrate, account for 60 % of the sphalerite in the Cu-concentrate. These losses are due to activation, or lack of selective flotation by the Minerec collector, and are not amenable to regrinding. Sphalerite losses to the tailings are largely as middling (20 %), and sub-middling (65 %) grains, and correspond to 10 % of the sphalerite in the feed. Furthermore, 8 % of the sphalerite in the feed is lost to the Cu-concentrate as middling and sub-middling grains. Thus, the loss of 18 % of the sphalerite is the maximum amount of sphalerite that is potentially amenable to extremely

fine regrinding, and improved recovery. However, the lack of suitable regrind streams limit the expected maximum improvement in sphalerite recovery to 10 %.

It is recommended that regrinding be coupled with increased 'pull' by the roughers and increased rejection by the cleaners to optimize concentration of middling and sub-middling grains of chalcopyrite and sphalerite in the respective cleaner tailings. The Cu-1st cleaner tailings or the coarse fraction (e.g. > 50 um) of the Cu-rougher concentrate are alternative streams recommended for fine regrinding (i.e. 80 % - 26 um). The Zn-1st cleaner tailings is recommended as a secondary regrind stream, principally for sphalerite liberation. Optimized fine regrinding has the potential of increasing chalcopyrite and sphalerite liberation and recovery by approximately 10 % each, corresponding to improved Cu and Zn recoveries of up to 8 and 6 %, respectively.

The probable principal hosts of Ag are freibergite, chalcopyrite, pyrite, sphalerite and mercurian electrum. Sphalerite accounts for a maximum of 8.5 % of the Ag in the ore. The probable principal hosts of Au are mercurian electrum and 'invisible Au' in pyrite and arsenopyrite. The high Au losses to the tailings (i.e. 46.3 %) probably reflect 'invisible Au' in pyrite and arsenopyrite, but also mercurian electrum locked in pyrite and arsenopyrite, due to the poor grind and low liberation.

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Chapter I

INTRODUCTION

A mineralogical study was conducted on mill products (i.e. concentrator slurry samples) from the HBMS concentrator circuit as a project within the CANADA-MANITOBA MINERAL DEVELOPMENT AGREEMENT. This project entitled "The mineralogical characteristics that affect metal recoveries from the Cu, Zn, Pb and Ag ores from Manitoba" was conducted as a contract to Hudson Bay Mining and Smelting Co. Ltd. (HBM&S) from CANMET, Dept. of Energy, Mines and Resources, under the supervision of the CANMET Scientific Authority Dr. W. Petruk. The work was performed by the author at the University of Manitoba as an employee of HBM&S under this contract.

The objective of the project was to establish whether the volcanogenic massive sulphide deposits of Manitoba contain unique mineralogical characteritics, and to determine whether these characteristics can be utilized to modify metallurgical practises, so that recovery could be optimised. Mineralogical studies on the ores from the Trout Lake deposit, and mill products from the Trout Lake concentrator circuit in Flin Flon comprised Parts 1 and 2 of the project. The Trout Lake deposit had been selected first for study because the copper concentrate commonly contains 5 Wt % zinc, and because approximately 30 % of the gold and of the silver are lost to the tailings. Part 1 sought to define the mineralogical characteristics that could have a bearing on mining and mineral processing (Healy & Petruk 1988), whilst Part 2 evaluated the behaviour of the minerals in the concentrator circuit (Healy & Petruk 1989).

Part 3 of the project involved sampling the HBMS concentrator circuit, which is also housed in the Flin Flon concentrator, and which for accounting reasons only, is operated separately from that of Trout Lake. Sieve size, chemical, petrographic and image analysis was done on each of the products. The image analysis included modal, grain size and mineral liberation analysis. Material balance calculations were done on several of the analytical data thus gathered, to determine the behaviour and inter-relationships of the metals and minerals through the circuit. The HBMS circuit poses a unique problem to the conventional method of evaluating circuit performance, in that the Cu and Zn assays are masked by a component of non-recoverable, recycled Cu and Zn in backfilled slag. This paper covers Part 3 of the project.

Chapter II

SAMPLING CAMPAIGN

A sampling campaign of the HBMS concentrator circuit was conducted on April 16th 1987. The strategy employed was to collect samples of the feed and all the discharge products (concentrates and tailings) from each unit (containing as many as 5 banks) of flotation cells in the entire cicuit (Petruk & Klymowsky 1977 Petruk & Schnarr 1979 and Healy & Petruk 1989). Due to poor access, samples of (1) the combined Cu-rougher feed, (2) the Cu-rougher tailings before regrinding, and (3) the combined Zn-1st cleaner feed were not collected (See Fig. 1). Conversely, for two of the products, several samples of essentially the same product were collected (e.g. 3 samples of the Cu-rougher concentrate) because access was only possible to multiple banks or streams. Thus, the 20 samples collected represent 17 products. Using a sampling team of 4 personnel, the samples were collected over a period of 2 hours. A cut of approximately 500 ml of slurry was taken at each sample point every 15 minutes to produce 4 to 5 litres of slurry per sample. The slurry was transferred from the sample pails to a pressure filter, and the filtered solids were oven dried. The dried samples were then quartered into subsamples, bagged and labelled for chemical analysis (1 quarter), sieve size analysis (2 quarters), and for preparation as polished sections (1 quarter). The samples were numbered such that the overflows and underflows were assigned even and odd numbers, respectively, and the numerical order approximated to their position through the circuit (See Fig. 1). The sample numbers are prefixed with 'HB.MP', denoting 'HBMS Mill Product'.

The on-stream x-ray analyzer indicated that relatively steady-state conditions prevailed during the sampling campaign, with only moderate changes in feed grades, but with more significant fluctuations in Cu and Zn recoveries in the end products (See Table 1). The on-stream analyzer also indicated that the grade of the feed was slightly lower in Cu and higher in Zn relative to the average yearly mill head grade (See Table 2). In addition, a 10 kg grab sample of the - 1.25 inch rod mill feed was collected 'off the belt' midway through the sampling campaign.

Ore from the Centennial mine comprised approximately 35 % of the feed, whereas the remaining 65 % consisted of ore from the South Main (Flin Flon) mine. Centennial ore is generally 'hard' and difficult to grind, whereas South Main ore is generally talcose and easy to grind.

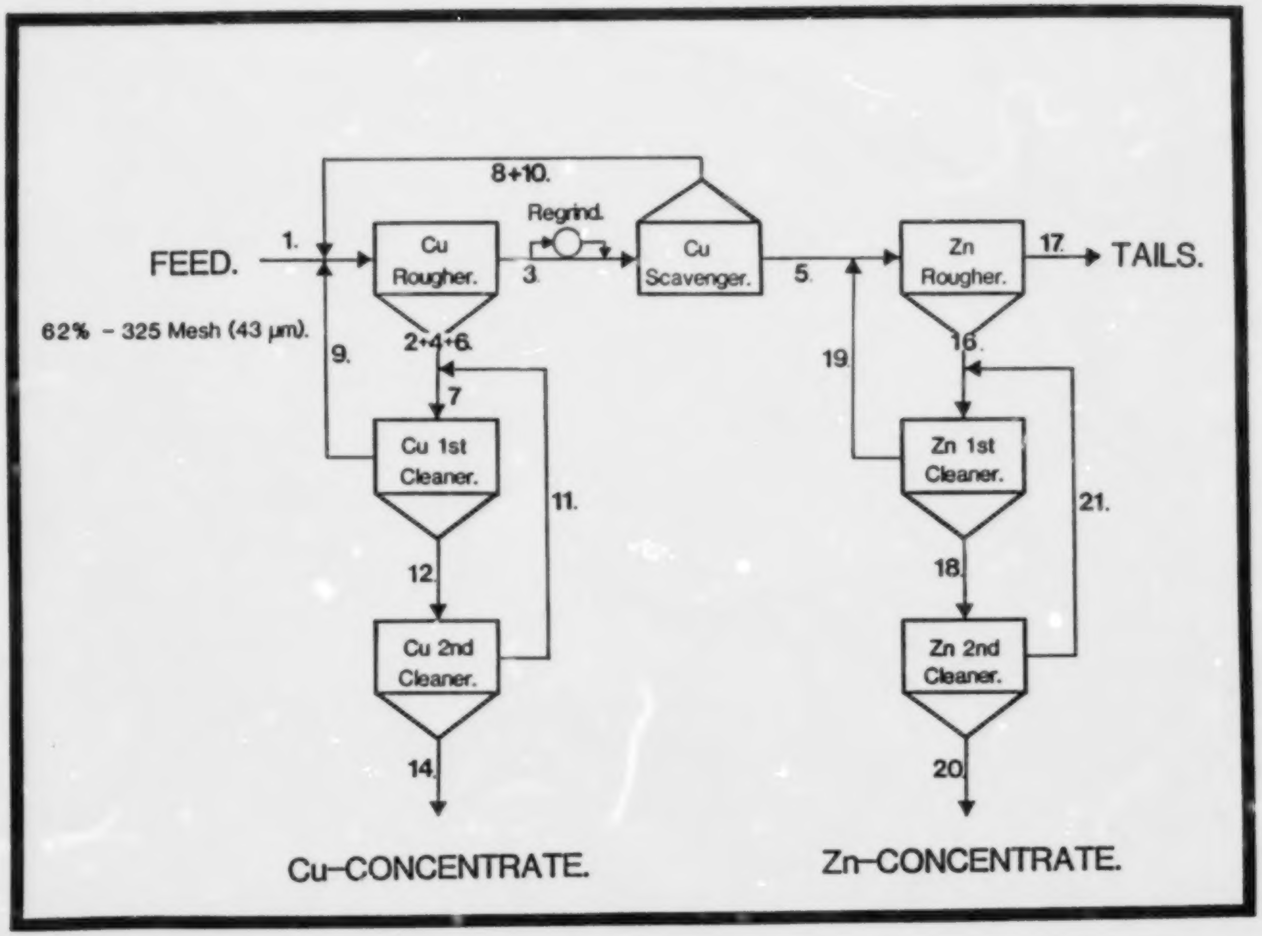
FEED.
1.
62% - 325 Mesh (43 µm).
Cu Rougher.
2+4+6.
3.
Regrind.
8+10.
Cu Scavenger.
5.
9.
7
Cu 1st Cleaner.
11.
12.
Cu 2nd Cleaner.
14.
Cu–CONCENTRATE.
Zn Rougher.
17.
TAILS.
16.
19.
Zn 1st Cleaner.
21.
18.
Zn 2nd Cleaner.
20.
Zn–CONCENTRATE.

FIGURE 1.
THE HBMS CONCENTRATOR CIRCUIT PORTRAYING THE 20 SAMPLE POINTS.

TABLE 1

STREAM ANALYSES BY ON-STREAM X-RAY ANALYZER DURING THE SAMPLING CAMPAIGN.

TIME	HEADS		Cu-CONCENTRATE				Zn-CONCENTRATE				TAILINGS	
	Cu	Zn	Cu	Cu-Rec	Zn	Zn-Rec	Cu	Cu-Rec	Zn	Zn-Rec	Cu	Zn
1.31	1.26	2.49	16.40	85.0	7.45	19.6	0.59	1.3	50.78	58.6	0.19	0.60
1.51	1.19	2.51	17.75	82.7	6.30	13.9	0.64	2.0	45.40	68.3	0.20	0.49
2.04	1.36	2.69	16.45	84.8	7.18	18.8	0.70	1.4	51.76	53.2	0.21	0.84
2.17	0.96	2.32	18.83	75.5	7.02	11.6	1.06	4.3	41.75	70.2	0.21	0.46
2.33	0.94	2.20	19.00	71.5	8.31	13.3	1.38	4.9	39.36	58.9	0.24	0.65
2.50	0.87	2.18	18.55	73.0	8.54	13.4	1.20	3.5	48.51	57.0	0.22	0.69
3.06	1.25	2.17	18.93	81.6	8.70	21.6	1.13	1.8	51.93	48.1	0.22	0.71
3.22	1.26	2.25	18.54	81.7	8.68	21.4	1.15	1.9	52.57	49.5	0.22	0.71
3.38	1.28	2.28	18.70	82.1	8.37	20.5	1.09	1.7	54.83	48.2	0.22	0.77

* Sampling campaign occurred precisely between 1.30 and 3.30 pm on April 16th 1987.

TABLE 2

HBMS MILL HEAD GRADES.

	Cu(%).	Zn(%).
1984 (Yearly).	1.75	2.17
1987 (Typical).	1.50	1.50
1987 (April - This Study).		
On-Stream X-ray Analyzer.	1.15	2.34
Bulk Chemical Analyses.	1.19	1.91

Chapter III

METHODS OF INVESTIGATION

CHEMICAL ANALYSIS

The twenty samples were analysed chemically for Cu, Zn, Au, Ag, Cd, Pb, Fe, S and Co at the HBM&S assay laboratory, Flin Flon, and for As, Se, Te, Hg, Sn and Sb by Bondar-Clegg, Ottawa. The dried samples required no preparation for chemical analysis, as they are fine-grained and homogeneous. The HBM&S assay laboratory and the Bondar-Clegg laboratory have previously achieved a relatively high degree of accuracy and precision (Healy & Petruk 1988, 1989), hence no reference ore standards were included with this batch of samples.

SIEVE SIZE ANALYSIS

The twenty samples were submitted to HBM&S for sieve size analysis. The samples were machine sieved, using stacked Tyler Series meshes with the mesh numbers 48, 65, 100, 150, 200 and 325. The weight of retained material in each sieve, and that passing 325 mesh was determined and recorded for each sample.

PETROGRAPHIC AND IMAGE ANALYSIS

The twenty samples were prepared as polished sections at CANMET, Ottawa. Approximately 200 mg of the sample was mounted in epoxy/araldite in such a manner as to minimize fractionation of the sample by gravity settling of the particles. The polished sections were petrographically examined to confirm comparable mineralogy to that at Trout Lake (Healy & Petruk 1988, 1989). During petrographic and image analysis of the polished sections, most of the ore minerals[1] at Trout Lake were observed (and confirmed by EDXA), except for dyscrasite, freieslebenite, boulangerite, pilsenite, volynskite, naummanite and acanthite. Furthermore, one grain of guanajuatite (Bi_2Se_3) was observed. The petrographic examination also qualitatively determined the abundance, median grain size, degree of liberation, and the type of locked species of the minerals of interest. This information was useful in constructing appropriate image analysis routines. Furthermore, the petrographic examination also revealed the abundance of Cu-Fe-sulfides (e.g. covellite, chalcocite, bornite-chalcocite, Iss) derived from entrained Cu-matte in backfilled Cu-smelter slag that is introduced into the ore feed by mine dilution (Healy & Petruk 1989).

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[1] Ore minerals are informally defined here as minerals characteristic of the ore zone, but are not necessarily of economic value e.g. sulfides, alloys, sulfosalts, selenides, tellurides, etc.

The image analysis was done at CANMET, Ottawa on the MP-SEM-IPS image analysis system, which was developed by Petruk (Petruk 1987). The SEM capabilities of the Jeol 733 electron microprobe, the image producing component of the system, permit the generation of high resolution backscattered electron (BSE) images of the polished sample surface. The BSE (or grey level) image of a single field of the sample surface is tranferred to the Kontron SEM-IPS image analyser and digitized. Those areas of the image detected within the grey level window of each mineral are transformed into separate binary images. These mineral-specific binary images are stored in the image memory assigned to each mineral or mineral group (i.e. gangue[2] or pyrite + marcasite). At this point the various stereological measurements are made, and the procedure is repeated for successive fields. For further details of the specific technique used, see Healy & Petruk (1987) and Petruk (1988).

The mineral quantitites were determined by measuring the relative areal fraction of each mineral (or group), and were normalized to area or volume %. Grains of the mineral(s) of interest were also described on the basis of their grain size distribution and degree of liberation i.e. classification. The grains sizes of chalcopyrite, sphalerite and galena were determined by measuring the surface areas of grains of the respective minerals, and using the square root of the area as the diameter. This 'equivalent square diameter' (or 'equivalent edge length of a square') gives grain sizes that compare closely with sieve size analyses (Petruk 1978).

The degree of liberation is measured using a classification, which describes the degree to which each particle containing a grain of the mineral of interest is locked with other minerals. Grains of the mineral of interest, and which comprise a variable volume % of the particle which they are identified as occurring in, are classified as being free (> 99.9 volume % of the particle) or unliberated (< 99.9 volume %). The grains are classified into an eleven-fold classification (i.e. free grains in the 99.9-100 class and ten classes of unliberated grains with intervals of 10 volume % each) based on the grains contribution in terms of volume % to the total volume of the particle. This classification is simplified into a four-fold classification based on the expected flotation characteristics of the particle. The 'free' grains are defined as comprising 99.9 to 100 % of the particle. The 'sub-free' particles are those in which the grain comprises between 99.9 % and 70 % of the particle, and which is expected to behave essentially as if free, i.e. in accordance with the flotation characteristics of the mineral of interest. The 'middling' particles are those in which the grain comprises 69.9 % to 30 % of the particle, and whose flotation characteristics are likely to mimic that of either the mineral of interest or the accompanying mineral(s). The 'sub-middling' particles are those in which the grain comprises less than 29.9 % of the particle, and whose flotation characteristics are expected to correspond to that of the dominant accompanying mineral(s).

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[2] Gangue is informally defined here as non-sulfide minerals characteristic of the surrounding hostrocks, and of no economic value i.e. silicates, carbonates and phosphates.

Polished sections of each product were analyzed twice; first by a routine dedicated to the major minerals, and secondly, by a routine dedicated to the minor minerals. The major minerals routine determined the following: (1) quantities of each mineral (or mineral group); (2) liberations (% free) of chalcopyrite, sphalerite and galena; (3) grain size distribution for the free chalcopyrite, sphalerite and galena; and (4) classification of free and unliberated chalcopyrite, sphalerite and galena. The minor minerals routine was employed to identify and measure the mineral quantities only of galena, freibergite, mercurian electrum (shortened for brevity to electrum in the text), hessite, pyrargyrite, gudmundite and other trace minerals of interest. These minerals are generally very fine-grained, present in trace amounts, and are difficult to identify because they cannot be differentiated from each other in the BSE image. These were identified by scanning particles that fall into a 'family' grey level window with the electron beam and determining their chemical composition by EDXA.

No stereological corrections were applied to the data to correct for stereological biases that operate on the dataset. Image analysis data are subject to biases produced by random sections of particles that preferentially generate (1) finer apparent grain sizes, and (2) higher apparent liberations. The first bias is minimized by determining the grain size diameter as the 'equivalent square diameter', which closely approximates to that determined by sieve sizing (Petruk 1978). The second bias, generated by sections of unliberated grains appearing as free grains, is generally smaller for 'natural' assemblages than predicted by theoretical models (Hill et al. 1986). Petruk (1978) empirically determined that less than 15 % of the unliberated grains appear as free grains in sections of mill products. Furthermore, Petruk has observed that raw data undergo smaller adjustments than stereologically corrected data when material balanced, supporting the contention that the stereological corrections applied, particularly to liberation data, are theoretically uncertain and inaccurate. In addition, the present data is material balance modelled and evaluated principally on the basis of intercomparison of samples, and hence mitigates the absolute-value imperative.

Image analysis using the major mineral routine was done on each of the polished sections corresponding to the 20 samples, whereas the minor mineral routine was not employed on all 20 samples, but only on 12 samples corresponding to 12 mill products. Sufficient additional samples were to be analyzed, but a decision to curtail the scope of the project pre-empted this work. The major mineral routine employed a magnification of x150 (pixel size = 0.936 um) over an analytical area of approximately 40 fields per sample. The minor mineral routine employed a magnification of x400 (pixel size = 0.368 um) over an analytical area of between 200 and 5,000 fields. An analysis by the major mineral routine of approximately 3,000 to 5,000 fields involved 2 overnight runs, and was required for those samples where grains of the minerals of interest are very rare.

Much of the data generated by the major mineral routine were summarized in tables, histograms and 3-D diagrams, and are compiled in Volume 2, some of which are presented in this volume.

MATERIAL BALANCE CALCULATIONS

In order to evaluate plant performance and accurately describe the steady-state operation of an ore processing circuit using quantitative measurements of selected parameters or process variables (e.g. chemical assays, or mineral liberations) from different streams in the circuit, it is necessary to do material balance calculations (Laguitton & Wilson 1979). The weight of the products are unknown, and the raw analytical data contains sampling (e.g. dynamic process fluctuations and fractionation during sampling), sample preparation and analytical errors. Thus, material balance calculations are used to generate a new set of coherent data, which may include otherwise inaccessible or experimentally undetermined data. The material balance calculations of this study were done using the MATBAL program, which is fully implemented on the WATFIV compiler on the University of Manitoba mainframe. The program employs the principle of mass balance, in which the feed to each bank of flotation cells is split into a concentrate/overflow and tailings/underflow, and the weight and assays (or any other measured variable) must balance (Petruk & Schnarr 1981). Because of the uncertainty in the raw data, the program uses a 'least squares refinement' technique to itteratively adjust the raw data, and produces 'best fit' or 'balanced' values.

The weight or flow rate of each stream is determined with the MATBAL program using chemical assays and mineral quantities as the most precise process variables available. Initially the stream flow rates were determined with the MATBAL program using the chemical assays for Au, Ag, Cu, Zn, Cd, Co, Fe, S and Pb, and the mineral quantities of gangue, pyrite and sphalerite. However, large adjustments in other variables were generated when material balance simulation of these process variables (e.g. mineral quantities and liberation data for sphalerite and chalcopyrite) were based on these flow rates. These adjustments were unacceptable as they resulted in significant cross-balancing errors (i.e. data is not globally coherent for different process variables), and particularly as these occurred for variables with satisfactory counting statistics (e.g. sphalerite mineral quantities). Several datasets containing different process variables were used to calculate flow rates, of which the dataset containing the 9 variables Ag, Cu, Zn, Fe, S assays, and the gangue, pyrite, sphalerite and chalcopyrite mineral quantities calculated the most acceptable flow rates. These flow rates were then fixed, and in turn generated the most globally coherent set of adjusted variables. The accepted flow rates of the Cu-concentrate, Zn-concentrate and tailings are 4.647, 1.565 and 93.79, respectively.

Having determined the stream flow rates for each product, these were than fixed, and other process variables were balanced by MATBAL to yield adjusted values of these variables. The MATBAL program simultaneously calculated the recoveries of each variable in each product. Material balances were done on several types of data in order to determine the recoveries of metals, minerals and of specific classes of free and/or unliberated grains. These data types are: (1) chemical assays; (2) mineral quantities (modal) data for all the minerals of interest; (3) modal data for free chalcopyrite and sphalerite (but not galena); and (4) modal data for composite classes of free plus unliberated chalcopyrite and sphalerite (but not galena).

Both the chemical assay and modal quantities data were material balanced by entering the raw data directly into the MATBAL program. However, the liberation data can only be material balanced after it is weighted in terms of the amount of the mineral(s) of interest present in each product. This is achieved by determining the modal fractions of the mineral of interest in each product that occur in the specific liberation categories to be material balanced (i.e. free, unliberated, sub-free, middling and sub-middling). This contrasts with the Trout Lake study (Healy & Petruk 1989), in which the Cu and Zn assays were taken as being the more precise modal equivalent of the chalcopyrite and sphalerite abundances, such that the modal fraction of the Cu assay, but not of chalcopyrite, occurring as free chalcopyrite (for example) was material balanced. This is the preferred procedure, as chemical assays are more precise than modal quantities generated by image analysis. In this study however, the Cu and Zn in slag mask the Cu and Zn assays due to chalcopyrite and sphalerite, respectively.

The modal quantities of a mineral of interest, chalcopyrite (for example) in each product are thus multiplied by that fraction of chalcopyrite that occurs in each of these composite classes. The modal fraction of chalcopyrite occurring as free grains (for example) in each product are then material balanced. The % liberation (i.e. % free) is the percent of the adjusted modal quantities of chalcopyrite occurring in the adjusted modal fraction of free chalcopyrite for a given product. Thus, a set of adjusted liberations are not generated by MATBAL, but are extracted from two sets of internally, but not necessarily globally, coherent data. Thus, it is stressed that MATBAL balances each set of process variables independently, such that the adjusted data may not cross-balance. However, ideally the closer the true values are approached for each variable, the closer the cross-balance achieved.

Chapter IV

NATURE OF THE FEED

The ore being processed at the time of the sampling campaign on April 16th was mined from the Centennial mine (approx. 35 %) and the South Main (Flin Flon) mine (approx. 65 %).

The composition of this float feed and unbalanced assays of the 1983 float feed are presented below in Table 3. It is apparent that the feed on April 16th contains essentially 'normal' Au, Ag, Zn, Pb, Fe and S contents, but relatively low Cu contents relative to 1983 feeds.

TABLE 3

CHEMICAL COMPOSITION OF HBMS FLOAT FEEDS.

Feed.	Au	Ag	Cu	Zn	Pb	Fe	S
	ppm.		Wt %.				
This Study.	1.235	15.09	1.19	1.91	0.084	19.3	12.4
1983.	1.303	16.46	1.75	2.17	0.100	21.9	11.1

Note: Assays are not material balance adjusted.

A 4.4 Kg split of the rod mill feed grab sample, which consists of less than 'one and a quarter inch particles', was hand sorted using visual discrimination, and assuming comparable ore types to those defined at Trout Lake (Healy & Petruk 1988, 1989). Particles less than 3.33 mm were too small to be sorted and were rejected using a Tyler series sieve (i.e. No. 6; - 3.327 mm). The results of the sorting are assumed to be true for this large (30.7 %) rejected fine-grained fraction. The Disseminated Pyrite + Chalcopyrite ore type was not picked out and probably contributed largely to the Chalcopyrite Stringer ore type and to a lesser extent to the Massive Pyrite ore type. The Sheared Chalcopyrite + Sphalerite ore type is difficult to identify in hand-specimen less than one and a quarter inches in diameter and was not sorted separately, and probably contributed to the Mixed/Banded Chalcopyrite + Sphalerite and the Chalcopyrite Stringer ore types. The Vein Quartz + Chalcopyrite ore type was not sorted, as particles consisting of coarse-grained, white quartz were not observed. Furthermore, backfilled slag was also sorted separately. The handpicked fractions were weighed and the results are presented in Table 4A.

The slag and waste fractions constitute a total of 43.6 Wt % of the feed. The slag comprise 9.35 %, whereas the waste fraction which consists of barren (less than 1 % visible sulfides) wallrock particles, comprises 34.2 % of the feed. Of the 43.6 % slag plus waste component, approximately 35 % (i.e. 15.3 %) is greater than 10 mm (See Table 4B). Thus, after correction for the rejected - 3.33 mm fraction, only approximately 11 % of the feed consists of slag and barren waste in the size range of greater than 1 cm; the fraction potentially amenable to pre-concentration by photometric ore sorting (Wills 1985). Thus, approximately 11 % of the feed on April 16th could potentially have been removed prior to grinding, with resultant decrease in grinding costs, reagent costs, and possibly talcose slime production. This small fraction of the feed that is potentially amenable to rejection is significantly less than at Trout Lake (i.e. 33 %), and indicates that pre-concentration is probably not a practical alternative for the HBMS circuit. The hydraulic backfill requirements for the 'Cut and Fill' mining method used at Trout Lake mitigates against the usefulness of pre-concentration. The rejected coarse waste fraction would have to be finely ground so as to supplement the main tailings, which are presently used for backfill purposes. The same consideration may also apply to the South Main and Centennial mines.

The hand picked fractions indicate that the principal ore type is the Chalcopyrite Stringer, with approximately equivalent amounts of the Banded Pyrite + Sphalerite and the Massive Pyrite ore types. The 18.1 % Massive Pyrite ore component may account for the poor grindability of the ore, while the 17.6 % Banded Pyrite + Sphalerite ore component contributes substantial pyrite, and probably also contributes to high pyrite recoveries in the Zn-concentrate, in the form of sphalerite-pyrite particles.

TABLE 4A

ORE TYPE ABUNDANCES IN THE HBMS ROD MILL FEED. DETERMINED BY HAND SORTING THE + 3.33 mm FRACTION.

FRACTION.	WEIGHT %.	NORMALISED WEIGHT %. (Waste-free)
Slag	9.35	----
Waste.	34.2	----
Cpy Stringer.	18.7	33.2
Massive Py.	18.1	32.1
Banded Py + Sphal.	17.6	31.2
Mixed/Banded Cpy + Sphal.	1.66	2.94
Massive Sphal.	0.38	0.67

TABLE 4B

SIEVE SIZES OF THE WASTE AND SLAG FRACTIONS OF THE HBMS ROD MILL FEED + 3.33 mm FRACTION.

TYLER SIEVE SIZES (mm)	WASTE.		SLAG.		WASTE + SLAG.	
	Wt.%	Cum.%	Wt.%	Cum.%	Wt.%	Cum.%
> 38.1	0.00	100.0	0.00	100.0	0.00	100.0
> 18.9	8.44	91.6	10.8	89.2	8.95	91.1
> 13.3	24.9	66.7	31.4	57.8	26.3	64.8
> 6.68	38.5	28.2	39.7	18.1	38.8	26.0
> 4.70	12.9	15.3	10.1	8.01	12.3	13.7
< 4.70	15.3	----	8.01	----	13.7	----
> 10 mm.	33 %		42 %		35 %	

Chapter V

CHEMICAL ANALYSIS

The twenty samples were chemically analysed for 15 elements, and the assays are presented in Table 5. The samples are ordered for presentation starting with the feed, the Cu-circuit concentrates, the Zn-circuit feed, the Zn-circuit concentrates, the Cu-circuit tailings, and the Zn-circuit tailings. This format will be consistently used throughout the report.

Two analyses for Sn and eight analyses for Te were below the detection limits 1 ppm. The remaining 13 elements gave concentrations for all samples above their respective detection limits. For the purpose of material balance calculations, half the value of the detection limit was substituted for data points that were below the detection limit (i.e. 'nd.'). This procedure provides relatively realistic first approximations to the true values, which the material balance adjustment is attempting to approach. Therefore, the adjusted assays and calculated recoveries for Sn and particularly Te contain significant uncertainties carried over from the raw data.

MATERIAL BALANCES OF THE CHEMICAL ANALYSES

The assays were adjusted by material balance calculations in order to remove analytical errors, and model the steady-state operation of the circuit. The adjusted assays and the calculated weights of each product are presented in Table 6. The recovery of the 15 elements in each of the 17 products was calculated simultaneously from these adjusted assays and are presented in Table 7.

The adjusted chemical assays of the 15 elements in the HBMS feed are compared with those in the Trout Lake feed (Healy & Petruk 1989) in Table 8. The HBMS feed has comparable levels of Cd and Te, depleted levels of Au, Cu, Zn, Hg, Se and Sn, and elevated levels of Ag, Pb, Fe, S, Co, As and Sb relative to that of Trout Lake. Compared to Trout Lake, Zn is proportionally more abundant than Cu in the HBMS feed, largely due to the 9.4 % slag component which contains approximately 6.5 % Zn and 1.5 % Cu. The higher S content indicates a higher total sulfide component in the ore, but which because of low Cu and Zn and higher Fe values, is principally reflected in increased amounts of pyrite. If the influence of the slag on the chemical assays was subtracted, and the values normalized on the basis of the total sulfur content, the ore would likely be seen to be extremely Fe-rich. Thus, the HBMS feed is expected to contain significantly higher amounts of pyrite, arsenopyrite, galena and possibly Sb-sulfosalts. This is consistent with increased proportions (relative to Trout Lake) of the Massive Pyrite and the Banded Pyrite + Sphalerite ore types in the hand sorted fractions of the rod mill feed.

Histograms of the recovery of the each element in each product are presented in Figure 2, and graphically describe the distribution and behaviour of each element through the circuit. Elements with similar distributions are readily grouped by simple visual comparison of these histograms. Four groups of elements are recognised.

Group 1 comprises the four elements Fe, Co, S and As, whose behaviour is essentially inert and report principally (> 80 %) to the tailings. Group 2 comprises the four elements Sb, Sn, Se and Pb, whose behaviour is partly inert with between 70 and 85 % of each of the elements reporting to the tailings. In addition, these elements exhibit comparable recoveries (between 15 and 25 %) to the Cu-concentrate, with minor recoveries (i.e. < 10 %) in the Zn-concentrate. Group 3 comprises the four elements Cu, Ag, Au, and Te, which display high recoveries (i.e. 65 - 95 %) in the Cu-rougher concentrates, and which are partly rejected through the Cu-cleaning circuit, principally in the Cu-1st cleaner. These elements exhibit low recoveries (i.e. < 10 %) in the Zn-circuit concentrates, but moderately high recoveries (i.e. 20 - 60 %) in the tailings. Group 4 comprises the three elements Zn, Cd and Hg, which exhibit similar behaviour in the Cu-circuit and are partly rejected through the Cu-cleaning circuit, resulting in minor recoveries (i.e. 15 - 22 %) in the Cu-concentrate. These elements exhibit comparable concentration in the tailings and in the Zn-circuit concentrates (i.e. recoveries between 26 and 52 %).

The circuit balance of the feed and the three main products for this study incorporates in tabular form, much of the information in these histograms, and is presented in Table 9. The most notable feature of Table 9 is that the Cu-concentrate assays 17.4 % Cu, and the Zn-concentrate assays 46.2 % Zn, constituting 51 % and 76 %, respectively of the maximum attainable grade. This is based on the composition of stoichiometric chalcopyrite (i.e 34.5 % Cu), and an estimated average composition (i.e. 61 % Zn) of Centennial sphalerite (i.e. 63.5 % Zn; See Scott 1977) and Flin Flon sphalerite (i.e. 59.2 % Zn; See Koo 1973). The 17.4 % Cu assay is significantly lower than the 28.0 % observed by Healy & Petruk (1989) in the Trout Lake Cu-concentrate, while the recovery of 77.5 % is also significantly worse than 91.7 % in the Trout Lake Cu-concentrate. The Zn recovery in the Zn-concentrate of 40.2 % is testimony to the 'inferior' Zn metallurgy of this circuit. The Cu-concentrate assays 7.52 % Zn, representing a 19.4 % Zn recovery, and is significantly worse than the 3.99 % Zn assay, corresponding to 11.0 % recovery in the Trout Lake Cu-concentrate (Healy & Petruk 1989). In the Trout Lake circuit, the principal object of investigation was resolution of the Zn recovery/loss to the Cu-concentrate. In this study, characterization of the Cu and Zn losses to the tailings, above those contributed by Cu- and Zn-bearing slag, are also significant components of the investigation. Similarly, resolution of the high loss of Au and Ag to the tailings (46.3 % and 32.4 %, respectively) are other primary objectives of this study.

The Group 1 elements Fe, Co, S and As exhibit high recoveries (i.e. > 83 %) in the tailings and reflect the behaviour of the inert minerals pyrite, pyrrhotite, arsenopyrite and Fe-bearing silicate gangue minerals (e.g. biotite). Pyrite is the principal host of Fe, Co and S, and is possibly a significant host of As, in addition to arsenopyrite. The low As recovery in the Zn-concentrate indicates that unlike at Trout Lake, disseminations of fine-grained arsenopyrite in the Banded Pyrite + Sphalerite ore type are minor

TABLE 5

THE RAW CHEMICAL ASSAYS FOR THE 20 HBMS MILL PRODUCT SAMPLES.

SAMPLE.	Au	Ag	Cu	Zn	Cd	Fe	S	Pb	Co	As	Te	Hq	Se	Sb	Sn
	(ppm)		(Wt %)								(ppm)				
1. Feed.	1.235	15.1	1.19	1.91	0.006	19.3	12.4	0.084	0.015	0.23	nd.	4.60	44.	43.	2.
8. Cu-Scav.Co.	5.557	47.3	5.70	5.30	0.016	23.9	24.1	0.240	0.028	0.99	3.	15.2	91.	120.	8.
10. Cu-Scav.Co.	5.900	65.5	5.40	4.80	0.014	24.1	23.5	0.260	0.029	0.41	3.	13.7	90.	110.	3.
2. Cu-Roug.Co.	7.134	95.0	8.40	5.80	0.017	26.8	28.4	0.270	0.030	0.38	3.	17.4	106.	127.	7.
4. Cu-Roug.Co.	7.820	97.8	9.10	5.60	0.015	26.1	27.3	0.240	0.027	0.47	3.	15.8	111.	126.	9.
6. Cu-Roug.Co.	6.380	85.4	7.00	6.00	0.017	26.5	27.7	0.260	0.031	0.45	3.	14.5	103.	120.	6.
7. Cu-1st Cl.Fd.	6.654	95.0	8.30	5.60	0.016	26.6	27.7	0.240	0.031	0.42	2.	16.2	103.	124.	10.
12. Cu-1st Cl.Co.	10.50	136.	13.4	7.50	0.022	27.7	31.9	0.320	0.031	0.54	4.	17.8	126.	163.	12.
14. Cu-2nd Cl.Co.	9.878	160.	18.0	7.40	0.021	27.5	32.8	0.320	0.026	0.47	4.	18.9	134.	188.	10.
15. Zn-Roug.Fd.	0.892	6.52	0.26	1.85	0.006	19.3	12.1	0.082	0.015	0.35	nd.	4.20	37.	31.	3.
16. Zn-Roug.Co.	2.950	46.3	1.14	30.0	0.080	19.0	31.8	0.420	0.016	0.65	nd.	48.6	57.	109.	13.
18. Zn-1st Cl.Co.	3.087	47.7	1.18	41.0	0.108	15.7	34.1	0.510	0.014	0.59	nd.	70.9	54.	114.	12.
20. Zn-2nd Cl.Co.	3.499	48.7	1.06	46.0	0.123	13.3	34.1	0.520	0.011	0.54	nd.	79.0	49.	120.	nd.
3. Cu-Roug.Tl.	1.098	9.26	0.28	1.62	0.005	19.2	11.6	0.077	0.015	0.27	nd.	3.70	37.	33.	6.
5. Cu-Scav.Tl.	1.098	8.58	0.23	1.57	0.005	18.9	11.5	0.074	0.014	0.24	nd.	4.00	34.	38.	3.
9. Cu-1st Cl.Tl.	3.704	40.1	2.10	4.50	0.013	25.9	24.7	0.200	0.032	0.49	2.	10.5	79.	77.	nd.
11. Cu-2nd Cl.Tl.	7.615	69.3	3.80	6.60	0.019	28.1	29.5	0.240	0.039	0.55	4.	16.4	89.	108.	6.
19. Zn-1st Cl.Tl.	3.293	38.4	1.14	8.70	0.024	26.9	28.8	0.320	0.024	1.10	2.	25.6	61.	99.	16.
21. Zn-2nd Cl.Tl.	3.842	63.5	1.35	23.0	0.062	24.1	34.9	0.510	0.020	0.82	1.	51.4	58.	117.	20.
17. Zn-Roug.Tl.	0.343	3.09	0.22	0.70	0.003	19.2	11.1	0.070	0.015	0.27	nd.	2.10	33.	25.	5.

Note : 'nd.' denotes not detected.

TABLE 6

THE ADJUSTED ASSAYS FOR THE 17 HBMS MILL PRODUCTS.

PRODUCT.	Au	Ag	Cu	Zn	Cd	Fe	S	Pb	Co	As	Te	Hg	Se	Sb	Sn
	(ppm)		(Wt %)								(ppm)				
1. Feed.	1.029	12.70	1.0	1.8	0.006	19.5	12.6	0.088	0.015	0.27	0.4	4.3	40.	38.	3.1
2. Cu-Scav.Co.	5.762	56.94	5.6	5.1	0.015	24.0	23.8	0.220	0.029	0.70	3.0	14.5	91.	115.	6.0
3. Cu-Roug.Co.	6.689	91.92	8.4	5.7	0.016	26.5	27.8	0.250	0.030	0.45	2.6	14.6	103.	122.	5.6
4. Cu-1st Cl.Fd.	6.757	89.87	8.0	5.8	0.017	26.7	28.0	0.250	0.031	0.46	2.7	14.7	102.	121.	5.6
5. Cu-1st Cl.Co.	10.33	148.5	14.9	7.4	0.021	27.6	32.0	0.310	0.029	0.48	3.8	19.0	128.	172.	11.1
6. Cu-2nd Cl.Co.	10.87	165.3	17.4	7.5	0.021	27.5	32.5	0.324	0.027	0.47	3.7	19.5	136.	186.	12.1
7. Zn-Roug.Fd.	0.583	5.488	0.3	1.6	0.005	19.2	11.8	0.078	0.015	0.27	0.3	3.8	36.	32.	2.8
8. Zn-Roug.Co.	3.224	45.96	1.1	32.7	0.085	18.5	32.1	0.437	0.016	0.71	0.4	56.3	55.	111.	7.6
9. Zn-1st Cl.Co.	3.327	52.14	1.1	42.4	0.110	15.4	34.1	0.506	0.013	0.57	0.3	70.2	52.	117.	4.7
10. Zn-2nd Cl.Co.	3.259	50.08	1.1	46.2	0.119	13.6	33.9	0.505	0.012	0.52	0.2	73.9	51.	118.	1.1
11. Cu-Roug.Co.	0.549	5.145	0.3	1.5	0.005	19.1	11.6	0.076	0.015	0.26	0.3	3.6	36.	31.	2.6
12. Cu-Scav.Tl.	0.549	5.145	0.3	1.5	0.005	19.1	11.6	0.076	0.015	0.26	0.3	3.6	36.	31.	2.6
13. Cu-1st Cl.Tl.	3.773	40.82	2.1	4.5	0.013	25.9	24.6	0.199	0.032	0.44	1.8	11.2	80.	78.	1.0
14. Cu-2nd Cl.Tl.	7.615	68.94	3.8	6.7	0.019	28.1	29.5	0.240	0.040	0.56	3.8	16.4	89.	108.	6.2
15. Zn-1st Cl.Tl.	3.190	45.28	1.2	8.7	0.025	27.2	28.7	0.317	0.024	1.04	0.7	25.0	62.	98.	19.0
16. Zn-2nd Cl.Tl.	3.773	62.08	1.4	22.9	0.062	24.2	34.9	0.511	0.020	0.83	0.9	51.5	58.	116.	23.3
17. Zn-Roug.Tl.	0.515	4.459	0.2	0.8	0.003	19.1	11.2	0.069	0.015	0.25	0.3	2.4	35.	30.	2.7

TABLE 7

THE ELEMENT RECOVERIES IN THE 17 HBMS MILL PRODUCTS.

PRODUCT.	Au	Ag	Cu	Zn	Cd	Fe	S	Pb	Co	As	Te	Hg	Se	Sb	Sn
1. Feed.	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0
2. Cu-Scav.Co.	0.98	0.80	0.94	0.49	0.45	0.22	0.33	0.44	0.33	0.46	1.25	0.59	0.39	0.52	0.34
3. Cu-Roug.Co.	73.3	83.0	91.0	36.0	31.7	15.5	25.1	32.3	22.2	19.2	69.0	38.3	29.1	36.1	20.6
4. Cu-1st Cl.Fd.	80.6	88.4	94.6	39.7	34.9	16.9	27.4	35.0	24.7	23.3	77.9	42.0	31.2	38.8	22.6
5. Cu-1st Cl.Co.	56.2	66.7	81.1	23.1	20.3	7.99	14.3	19.9	10.8	10.2	50.1	24.7	17.9	25.3	20.3
6. Cu-2nd Cl.Co.	49.0	61.3	77.5	19.4	17.1	6.57	12.0	17.2	8.23	8.15	41.2	21.0	15.7	22.5	18.4
7. Zn-Roug.Fd.	53.7	41.4	23.4	84.8	86.6	94.7	90.0	85.9	93.2	95.3	60.2	84.0	85.6	79.7	87.1
8. Zn-Roug.Co.	7.61	8.92	2.61	44.1	35.4	2.31	6.20	12.1	2.56	6.46	2.21	31.6	3.35	7.06	6.02
9. Zn-1st Cl.Co.	6.01	7.72	2.04	43.7	34.9	1.47	5.04	10.7	1.58	3.99	1.48	30.2	2.43	5.72	2.86
10. Zn-2nd Cl.Co.	4.90	6.22	1.64	40.2	31.7	1.09	4.20	8.95	1.18	3.05	0.81	26.6	1.99	4.79	0.54
11. Cu-Roug.Tl.	52.0	39.5	23.4	81.1	83.4	93.6	88.3	83.3	92.1	92.3	60.0	79.6	84.7	78.0	82.0
12. Cu-Scav.Tl.	50.9	38.7	22.5	80.6	82.8	93.4	88.0	82.7	91.8	91.9	58.8	79.0	84.3	77.5	81.7
13. Cu-1st Cl.Tl.	24.3	21.7	13.5	16.6	14.6	8.88	13.1	15.2	13.9	11.1	27.7	17.3	13.3	13.6	2.27
14. Cu-2nd Cl.Tl.	7.25	5.40	3.57	3.63	3.22	1.42	2.30	2.69	2.55	2.04	8.91	3.72	2.17	2.75	1.97
15. Zn-1st Cl.Tl.	2.71	2.71	0.97	4.21	3.74	1.22	2.00	3.16	1.38	3.42	1.40	5.05	1.36	2.27	5.48
16. Zn-2nd Cl.Tl.	1.11	1.51	0.40	3.87	3.23	0.38	0.84	1.77	0.41	0.94	0.67	3.62	0.44	0.93	2.32
17. Zn-Roug.Tl.	46.3	32.5	20.8	40.5	51.2	92.3	83.8	73.8	90.6	88.8	58.0	52.4	82.3	72.7	81.1

TABLE 8

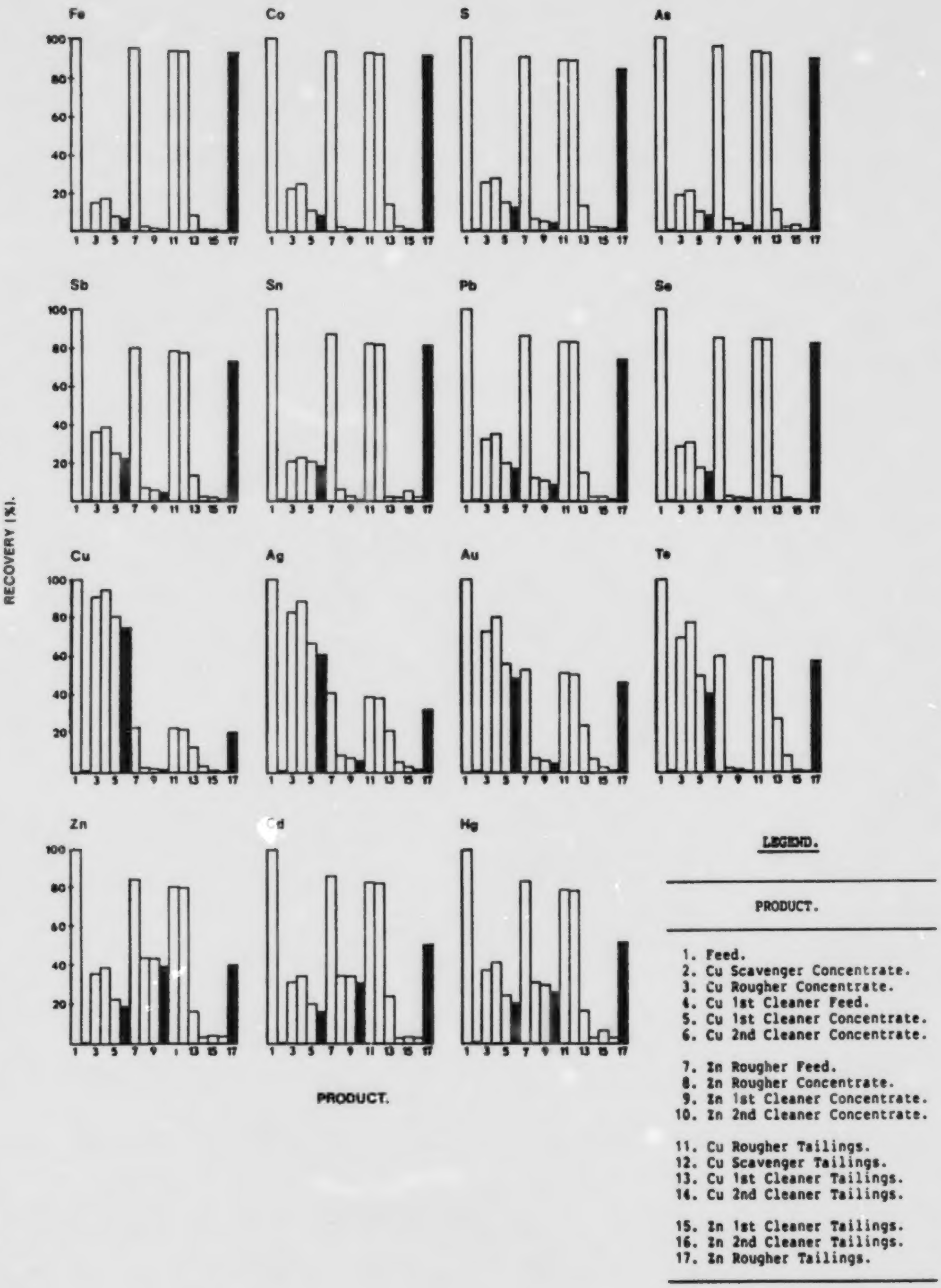
COMPARISON OF THE CHEMICAL COMPOSITION OF THE HBMS AND TROUT LAKE CIRCUIT FEEDS.

ELEMENT	HBMS	TROUT LAKE	ELEMENT	HBMS	TROUT LAKE
Fe (Wt%)	19.5	16.0	Se (ppm)	40.	49.
S (Wt%)	12.6	9.58	Sb (ppm)	38.	34.
Zn (Wt%)	1.80	2.57	Ag (ppm)	12.5	12.4
Cu (Wt%)	1.04	2.16	Hg (ppm)	4.	19.
As (Wt%)	.267	.075	Sn (ppm)	3.	7.
Pb (Wt%)	.088	.038	Au (ppm)	1.03	2.23
Co (Wt%)	.015	.010	Te (ppm)	.4	.4
Cd (Wt%)	.006	.006			

Note : Presented in order of decreasing abundance.
Trout Lake data from Healy & Petruk (1989).

in the ore, and do not contribute abundant sphalerite-arsenopyrite composite particles that may partly report to the Zn-concentrate.

The Group 2 elements Sb, Sn, Pb and Se report principally (i.e. > 72 and < 83 %) to the tailings, with moderate recoveries (i.e. 15 to 23 %) in the Cu-concentrate and low recoveries (i.e. < 10 %) in the Zn-concentrate. With the possible exception of Se, these elements exhibit significantly different behaviour than exhibited in the Trout Lake circuit. The 4.79 % recovery of Sb in the Zn-concentrate indicates that Sb-sulfsalts, principally freibergite, gudmundite and pyrargyrite, only rarely form composite particles with sphalerite, but report principally to the tailings, most probably locked with inert pyrite and/or galena. It is noteworthy that several free grains of freibergite were observed in the Cu-concentrate. The 81.1 % recovery of Sn in the tailings indicates that cassiterite, which was frequently identified during image analysis, is the principal host of Sn. Thus, unlike at Trout Lake, the interpreted Sn in solid solution in chalcopyrite (which will largely report to the Cu-concentrate), comprises a minor fraction (i.e. < 15 %) of the Sn assay. In contrast to the 45.5 % recovery of Pb in the Trout Lake Zn-concentrate, only 8.95 % is recovered in the HBMS Zn-concentrate, with 73.8 % reporting to the tailings. This indicates that galena is largely locked with pyrite, and/or as suggested by Healy & Petruk (1989), is partly inert to the Minerec collector. The somewhat higher recovery of Pb in the Zn-concentrate



LEGEND.

PRODUCT.

1. Feed.
2. Cu Scavenger Concentrate.
3. Cu Rougher Concentrate.
4. Cu 1st Cleaner Feed.
5. Cu 1st Cleaner Concentrate.
6. Cu 2nd Cleaner Concentrate.

7. Zn Rougher Feed.
8. Zn Rougher Concentrate.
9. Zn 1st Cleaner Concentrate.
10. Zn 2nd Cleaner Concentrate.

11. Cu Rougher Tailings.
12. Cu Scavenger Tailings.
13. Cu 1st Cleaner Tailings.
14. Cu 2nd Cleaner Tailings.

15. Zn 1st Cleaner Tailings.
16. Zn 2nd Cleaner Tailings.
17. Zn Rougher Tailings.

FIGURE 2.
HISTOGRAMS GIVING THE DISTRIBUTION OF THE RECOVERIES CALCULATED BY MATBAL FOR EACH ELEMENT IN THE 17 HBMS MILL PRODUCTS.

(i.e. 8.95 versus < 5 % for Sb, Sn and Se) is consistent with an association of fine-grained galena with sphalerite, and consequently forming composite particles with the latter in the ground products. As in the Trout Lake circuit, Se reports principally to the tailings (i.e. 82.3 %) with 15.7 % reporting to the Cu-concentrate, reflecting the behaviour of the principal probable hosts pyrite, chalcopyrite, pyrrhotite and clausthalite.

The group 3 elements Cu, Ag, Au and Te report principally (i.e. between 78 and 41 %) to the Cu-concentrate, with minor recoveries in the Zn-concentrate (i.e. between 0.8 and 6.2 %), and significant recoveries (i.e. between 32 and 58 %) in the tailings. As at Trout Lake, Cu exhibits very low recoveries to the Zn-concentrate, but in contrast, exhibits 20.8 % losses to the tailings. These high Cu losses reflect Cu in the slag silicate phases glass and fayalite (See COVELLITE AND CHALCOCITE), as well as unliberated chalcopyrite.

As at Trout Lake, approximately one third of the Ag is lost to the tailings (i.e. 32.9 %), but in contrast, only very minor (i.e. 6.22 %) losses occur to the Zn-concentrate, with 61.3 % of the Ag being recovered in the Cu-concentrate. In that 71 % of the sphalerite is recovered in the Zn-concentrate (See MINERAL QUANTITIES ANALYSIS AND MINERAL BALANCES), yet only 6.22 % of the Ag is recovered in the Zn-concentrate, sphalerite accounts for a maximum of 8.5 % of the Ag in the feed. This is significantly less than at Trout Lake (i.e. 19 %), is reflected in the higher Ag recovery in the Cu-concentrate, and suggests that freibergite, chalcopyrite and pyrite are the principal Ag-bearing minerals.

The recoveries of Au to the Cu-concentrate (i.e. 49.0 %) and to the Zn-concentrate (i.e. 4.90 %) are each approximately 10 % lower than at Trout Lake, and result in significantly higher losses to the tailings (i.e. 46.3 versus 25.5 %). Mercurian electrum is the sole Au-bearing mineral positively identified (petrographically and by EDXA during image analysis) in the HBMS mill products. It is expected to float into the Cu-concentrate if free, or if unliberated and locked with chalcopyrite. However, unliberated electrum locked with pyrite or arsenopyrite, and 'invisible' Au in pyrite and arsenopyrite, as observed at Trout Lake by Healy & Petruk (1990), are expected to be recovered in the tailings. The higher pyrite content of the HBMS feed (i.e. 15.4 versus 9.61 % pyrite; See MINERAL QUANTITIES ANALYSIS AND MINERAL BALANCES), and the coarser grind (i.e. K_{80} = 110 um versus 56 um) probably contributes to a higher fraction of unrecovered, unliberated electrum and 'invisible' Au. The lower Au recovery in the Zn-concentrate may reflect lower 'invisible' Au and unliberated electrum locked with pyrite, due to lower recoveries of pyrite and arsenopyrite in the Zn-concentrate.

As at Trout Lake, 0.8 % of the Te is recovered in the Zn-concentrate. However, while 80.6 % and 18.6 % of the Te is recovered in the Trout Lake Cu-concentrate and tailings, respectively, 41.2 % and 58.0 % of the Te is recovered in the HBMS Cu-concentrate and tailings, respectively. Hessite and rucklidgeite are the only Te-bearing minerals observed, and only in the Cu-circuit products. Possible Te in solid solution in chalcopyrite may also contribute to the Te assay in the Cu-concentrate, but significantly less than at Trout Lake, where chalcopyrite is interpreted to be the principal Te-bearing mineral (Healy & Petruk 1989). The principal host of Te, the least abundant of the 15 elements, and which might account for the high recoveries to the tailings, is not apparent.

The group 4 elements Zn, Cd and Hg exhibit between 40 and 53 % losses to the tailings, and unlike the previous three element groups, exhibit higher recoveries in the Zn- rather than Cu-concentrates. The high Zn losses of 40.5 % to the tailings and 19.4 % to the Cu-concentrate is in sharp contrast to the losses of 6.57 and 11.0 %, respectively in the Trout Lake circuit. However, this seemingly inferior Zn metallurgy includes the unrecoverable Zn in backfilled slag. This grades approximately 6.5 % Zn, having been produced prior to installation of the fuming plant. The Zn occurs principally in the slag silicates, but also in minor Fe-rich sphalerite associated with entrained Cu-matte phases. Thus, much of the Zn lost to the tailings may represent unrecoverable, 'recycled' Zn in slag silicates, as well as unliberated sphalerite. The loss of 19.4 % Zn to the Cu-concentrate is probably the result of a combination of (1) poor selective flotation by the Minerec collector; (2) activation by slag derived covellite and chalcocite; and (3) unliberated sphalerite locked with chalcopyrite.

The behaviour of Cd is similar, but not identical, to that of Zn. The discrepancy is interpreted to reflect the higher analytical uncertainties in the Cd determinations. Sphalerite is interpreted to be the sole Cd-bearing mineral in the ore (but possibly not in the feed). The distribution of Hg is virtually identical to that of Cd, suggesting that sphalerite is the principal host of Hg, with additional contributions from (1) Hg in electrum; (2) possible Hg in solid solution in pyrite and/or arsenopyrite; and (3) possible Hg in slag silicates. Both Cd and Hg may have higher solid solubilities in the slag silicates than Zn, which may be preferentially incorporated in the Cu-matte phases. As these silicates report principally to the tailings, higher losses of Cd and Hg to the tailings may be thereby generated. Nonetheless, in contrast to Trout Lake, where sphalerite accounts for less than approximately 25 % of the Hg in the ore, sphalerite accounts for between 40 and 80 % of the Hg in the HBMS feed, and perhaps as much as 90 % of the Hg in the undiluted (by slag) ore. This is consistent with sphalerite being the principal Hg-carrier in the Flin Flon orebody (T. Chen, pers. comm., 1986).

TABLE 9

THE HBMS CIRCUIT BALANCE ON APRIL 16th 1987.

PRODUCT.	PRODUCT WEIGHT.	ASSAYS.														
		Au	Ag	Cu	Zn	Cd	Fe	S	Pb	Co	As	Te	Hg	Se	Sb	Sn
	(Wt %)	(ppm)						(Wt %)						(ppm)		
Feed.	100.000	1.030	12.52	1.0	1.8	0.006	19.5	12.6	0.089	0.015	0.27	0.4	4.3	40.	38.	3.1
Cu-Conc.	4.647	12.73	165.2	17.4	7.5	0.021	27.5	32.5	0.324	0.027	0.47	3.7	19.5	136.	186.	12.1
Zn-Conc.	1.565	3.255	50.15	1.1	46.2	0.119	13.6	33.9	0.505	0.012	0.52	0.2	73.9	51.	118.	1.1
Tails.	93.787	0.508	4.329	0.2	0.8	0.003	19.1	11.2	0.069	0.015	0.25	0.3	2.4	35.	30.	2.7
								RECOVERY (%)								
Cu-Conc.		49.0	61.3	77.5	19.4	17.1	6.57	12.0	17.2	8.23	8.15	41.2	21.0	15.7	22.5	18.4
Zn-Conc.		4.94	6.28	1.64	40.2	31.9	1.09	4.20	9.04	1.18	3.05	0.81	26.6	1.99	4.79	0.54
Tails.		46.1	32.4	20.8	40.5	51.0	92.3	83.8	73.8	90.6	88.8	58.0	52.4	82.3	72.7	81.1

CONCENTRATION FACTORS

As the assays and recovery values represent non-normally distributed variables this dataset is not amenable to standard statistical techniques. A technique for comparing assay values in different products which involves calculation of 'Concentration Factors' (Petruk & Schnarr 1981) was used (See also Healy & Petruk 1989). The concentration factor compares the concentration of each element to that of every other element in that sample, as well as to that of the feed. This factor is therefore the relative concentration of an element compared to that of the feed. A concentration factor (CF) of 1 indicates no relative concentration, while a concentration factor of greater than one indicates relative concentration, and a concentration factor of less than one indicates relative depletion.

The FORTRAN 77 program CONFAC (Healy & Petruk 1989) was used calculate the concentration factors for the 15 elements. The concentration factors, which are given in Table 10 were plotted to produce discontinuous curves or each element from the Cu-circuit concentrates, to the Zn-circuit concentrates, to the Cu-circuit tailings, to the Zn-circuit tailings (See Fig. 3).

The concentration factors were determined for 14 of the same elements examined at Trout Lake; S being included in the HBMS dataset, and Mn and In being included in the Trout Lake dataset (See Healy & Petruk 1989). While concentration factors for different datasets are inequivalent, they can be compared in a qualitative manner. The most notable distinction between the above two sets of concentration factors is the generally lower magnitude of relative concentration for most elements concentrated in the HBMS Cu-concentrate, particularly Cu, Ag and Au (i.e. CF's < 5.05), yet the slightly higher relative concentration of Zn and Cd in the HBMS Zn-concentrate. This probably reflects the generally lower grade of the HBMS Cu-concentrate, resulting from the poor liberation and the need for the Cu-circuit to 'pull hard'. The concentration factors for Zn and Cd in the HBMS Zn-concentrate are therefore higher by comparison (CF's are calculated by comparison of the different elements' concentrations in the dataset), despite the actually lower recoveries of Zn and Cd in the HBMS Zn-concentrate (i.e. 40.2 and 31.9 %) compared to those in the Trout Lake Zn-concentrate (i.e. 82.5 and 74.0 %). Similarly, the concentration factors for Fe and Co in the HBMS tailings (i.e. CF's = 1.75 and 1.72) are lower than those in the Trout Lake tailings (i.e. CF's = 4.07 and 4.32). This results from the high losses of all elements to the HBMS tailings (i.e. all element recoveries in tailings > 20.8 %), despite higher rejection of Fe to the HBMS tailings (i.e. 92.3 %) compared the Trout Lake tailings (i.e. 83.3 %).

Ideally, the Cu curves show the behaviour of elements that are concentrated in the Cu-concentrate; Zn curves the behaviour of elements that are concentrated in the Zn-concentrate; and Fe curves the behaviour of those rejected to the tailings. Comparison of curves for other elements to the Cu, Zn and Fe curves indicate the tendency of other elements to concentrate in the Cu-concentrate, Zn-concentrate and tailings, respectively. Because significant amounts of Zn and Cu occur in the slag silicates, which together with some unliberated chalcopyrite and sphalerite report to the tailings, and because significant amounts of Zn are recovered in the Cu-concentrate, the curves for Cu and particularly Zn are not truely representative of elements that report wholly to the Cu- and Zn-concentrates, respectively. Similarly, but to a much lesser extent, the Fe content of chalcopyrite and sphalerite, and the recoveries of unliberated pyrite in the Cu- and Zn-concentrates, result in the Fe curves representing an element that is principally, but not entirely recovered (i.e. 92.3 %) in the tailings. Nonetheless, the Cu, Zn and Fe curves correspond to those of elements exhibiting the maximum relative concentration in the respective end products. Excluding the two intermediate streams of the Cu-2nd cleaner tailings and the Zn-2nd cleaner tailings, in which Te and Sn exhibit maximum relative concentration, respectively, Cu, Ag, Au and Te exhibit maximum relative concentration in the Cu-concentrate; Zn, Cd, S, Pb, Hg and Sb in the Zn-concentrate; and Fe, Co, As, Se and Sn in the tailings.

The Zn curves show an unusual behaviour in the Cu-circuit, though to a lesser extent than at Trout Lake. Both Zn and Fe should be rejected from the Cu-circuit, and hence their curves in the Cu-circuit should be similar. While Fe shows the strongest relative depletion of all 15 elements, Zn shows low relative concentration. Furthermore, there is a slight but progressive increase in concentration of Zn in the Cu-circuit concentrates (i.e. C.F.'s = 1.08 to 1.20), suggesting that sphalerite is preferentially floated over gangue, pyrite and arsenopyrite. It is also observed that the moderately low relative concentration of Zn through the Cu-circuit is the same as that for Cd and Hg. The Zn, Cd and Hg curves are virtually identical, and thus tend to confirm that sphalerite is the principal host of Cd and Hg.

Despite the low recoveries of Zn, Cd and Hg in the Zn-concentrate (i.e. 40.2, 31.7 and 26.6 %, respectively) high relative concentration of Zn, Cd and Hg (i.e. CF's = 19.3, 14.9 and 13.0, respectively) are observed, reflecting the low recoveries (i.e. < 9 %) of all other elements in the Zn-concentrate. These 3 elements also exhibit relative concentration in the Zn-1st cleaner tailings, where in the absence of a Zn-scavenger, rejection in the Zn-circuit is intensified. This pattern is also shown to as lesser extent by Sn, Pb, Ag and Au. This suggests minor circulating loads of cassiterite-sphalerite, galena-sphalerite, electrum-sphalerite, freibergite±pyrargyrite-sphalerite composite particles, where the latter types of particles and sphalerite grains (i.e. Ag solid solution in sphalerite) are all Ag-bearing.

The Fe curves are similar to those of Co, with the exception of in the Cu-cleaning circuit. Both Fe and Co exhibit moderate relative depletion in these concentrates, and together with As exhibit the lowest relative depletion in the Cu-concentrate (i.e. CF's = 0.41 to 0.52). As Fe is principally hosted in pyrite and gangue (and slag) silicates, and Co in pyrite and pyrrhotite, where these are expected to be rejected to the tailings, these elements show the strongest relative concentration of all the 15 elements in the tailings (i.e. CF's = 1.72 and 1.75).

The As and S curves differ substantially from those of Fe (and Co) in the Zn-circuit concentrates only. Both As and S show moderate relative concentration in the Zn-circuit concentrates, and also in the Zn-rougher tailings. Sulfur being essentially the sole anion of the sulfides reports to all products in proportion to their sulfide content and product weight. Pyrite (host of approximately 85 % of the S) and arsenopyrite, which are the principal hosts of As, report largely to the tailings. Thus, S and As show relative concentration in the tailings. However, as all elements show > 20 % recoveries in the tailings, the recoveries of 83.7 % of the S and 88.8 % of the As to the tailings translate into only moderate relative concentration (i.e. CF's = 1.56 and 1.62). Despite a recovery of only 4.20 % of the S in the Zn-concentrate, the highest relative concentration shown by S occurs here (i.e. CF = 2.03). This reflects the restricted chemistry of the Zn-concentrate, with low recoveries of most elements (i.e. < 9 %, except for Zn, Cd and Hg), and yet a high S assay (i.e. 33.9 % S) due principally to S in sphalerite. The low relative concentration of As in the Zn-concentrate (i.e. CF = 1.45) reflects minor recovery of As in arsenopyrite-sphalerite particles.

Gold exhibits relative concentration in both the Cu- and Zn-concentrates (i.e. CF's = 3.05 and 2.39, respectively) and minor relative depletion in the tailings (i.e. CF = 0.88). The Au concentration curves do not correspond to those of any other element, though they closely resemble those of Ag, and partly those of Cu. Au shows strongest relative concentration in the Cu-concentrate (i.e. CF = 3.05), and unlike in the Trout Lake circuit, is not enriched in the Cu-scavenger concentrate. The Au curves probably reflect (1) the recovery in the Cu-circuit concentrates of free electrum and electrum locked with chalcopyrite; (2) the recovery in the Zn-circuit concentrates of electrum locked in composite particles with sphalerite, pyrite and arsenopyrite; and (3) the recovery in the tailings of fine-grained inclusions entrapped in pyrite, but principally 'invisible' Au in pyrite and arsenopyrite. The high relative concentration of Au in the Cu-cleaner tailings indicates the rejection of high proportion of Au-bearing particles. These possibly correspond to coarse-grained free and unliberated electrum rejected by the Cu-cleaners, resulting in high circulating loads of Au in the Cu-circuit. As electrum has a mean specific gravity of approximately 15, the probable maximum grain size for optimum flotation is possibly only 30 to 40 um. It is probable that the poorer grind (and hence liberation of electrum), and the higher pyrite and arsenopyrite content (and hence probably higher 'invisible' Au content), contribute to the lower magnitude of relative concentration of Au in the Cu- and Zn-concentrates, as compared to those of Trout Lake (See Healy & Petruk 1989).

The Ag curve for the Cu-circuit concentrates most closely resembles that of Cu, and for the Zn-circuit concentrates resembles that of Zn, though the magnitude of relative concentration of Ag is less than that of Cu, and particularly Zn in the respective circuit concentrates. This suggests that Ag is hosted in chalcopyrite (and freibergite and electrum, which have similar flotation characteristics to chalcopyrite) and to a lesser extent in sphalerite. Unlike in the Trout Lake circuit, Ag does not exhibit relative concentration in all 17 products, probably reflecting simpler Ag-mineralogy (i.e. freibergite, chalcopyrite, pyrite, electrum and sphalerite probably account for the bulk of the Ag assay in the feed). Only Cu exhibits greater relative depletion in the tailings than Ag (i.e. CF's = 0.35 and 0.62, respectively), reflecting the two elements with the smallest losses to the tailings (i.e. 20.8 and 32.4 %, respectively). Much of the Ag lost to the tailings is probably in solid solution in pyrite.

The curves for Te are unlike those of any other element, showing strong relative concentration in the Cu-circuit concentrates, but with high rejection into the Cu-2nd cleaner tailings, and the Zn-rougher tailings, and with strong relative depletion in the Zn-circuit concentrates. In addition, Te shows the strongest relative concentration of all elements in the Cu-scavenger concentrate (i.e. CF = 2.85). This suggests that Te-bearing minerals, or host particles of the latter, are marginally floatable (i.e. float in the Cu-scavenger, but rejected by the Cu-cleaners). It is concluded that hessite and rucklidgeite float into the Cu-concentrate, when free or locked with chalcopyrite, but into the tailings when unliberated and locked with pyrite. Furthermore, possible Te in solid solution in pyrite probably comprises the bulk of the Te in the ore.

The concentration factor curves for Pb show an enrichment in the Zn-circuit concentrates and tailings, particularly the Zn-2nd cleaner tailings. The Pb curves correspond to the Zn curves, except for a much lower magnitude of relative concentration, and in contrast show relative concentration in the tailings. The Pb is due to galena, both free, and unliberated grains locked with sphalerite and pyrite. The combination of locked galena grains in particles containing these minerals, and the partly inert flotability of galena obscures the relationship to Zn and Fe.

The concentration factor curves for Sb are unlike those of Cu or Fe, and are weakly similar to those of Zn, and in part Pb and Ag. The moderate coupling in the behaviour of Sb, Zn, Pb and Ag suggests the presence of Ag in sphalerite and galena, and composite particles of freibergite ± gudmundite ± galena ± sphalerite. The moderate relative concentration of Sb in the Cu-concentrate (i.e. CF = 1.41) probably reflects the recovery of free and unliberated grains of freibergite, which were observed in the Cu-concentrate. There is moderate, but progressively increasing, relative concentration of Sb in the Zn-circuit concentrates (i.e. CF's = 1.17 to 2.34), coupled with low rejection to the Zn-2nd cleaner tailings, but high rejection to the Zn-rougher tailings. This suggests the recovery of freibergite ± gudmundite ± sphalerite composite particles in the Zn-concentrate, but direct rejection to the tailings of free gudmundite, and/or unliberated freibergite ± gudmundite locked with pyrite and/or galena.

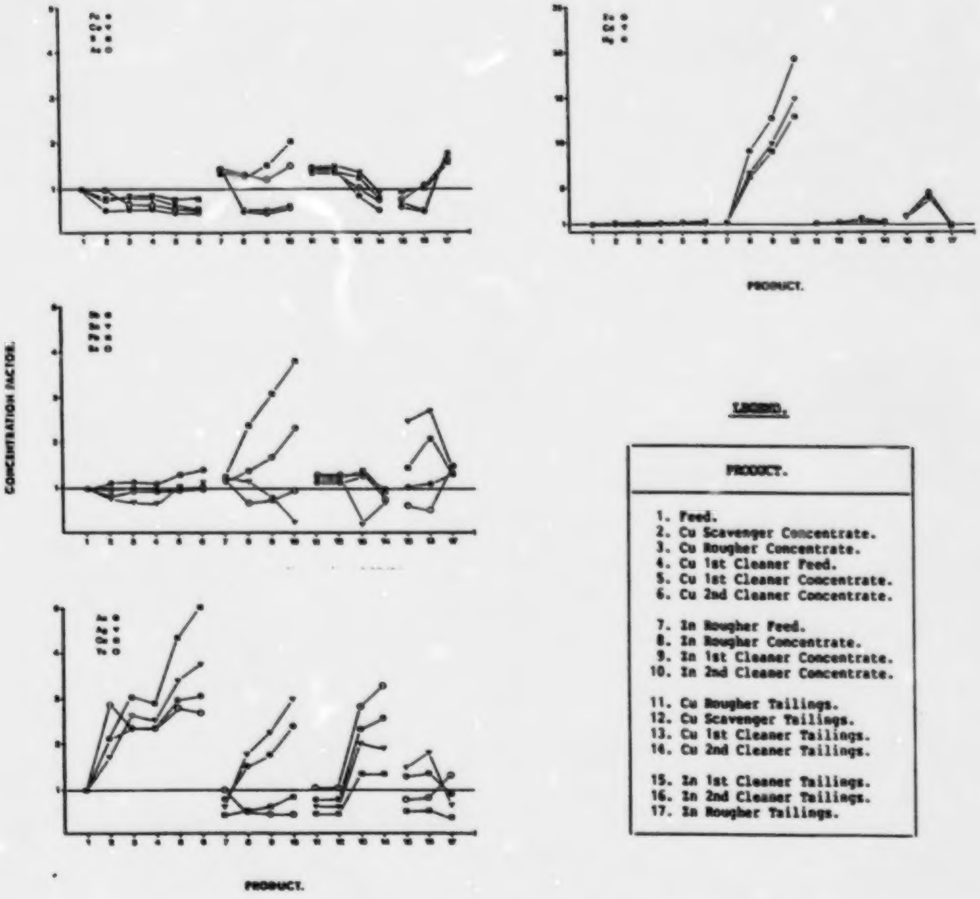
The concentration factor curves for Sn and Se are unlike those of Cu and Zn, but are weakly similar to those of Fe, all having high recoveries in the tailings (i.e. between 81 and 92 %). Sn shows relative concentration in the Zn-rougher concentrates, followed by high rejection into the Zn-1st and 2nd cleaner tailings, resulting in the lowest relative concentration of all elements in the Zn-concentrate (i.e. CF = 0.27). In contrast to the relative depletion of Fe in the Cu- and Zn-concentrates, the Se curves show neither relative concentration nor depletion. The principal hosts of Se are pyrite, pyrrhotite and possibly galena, all of which are partly recovered in the Zn-concentrate, but which report principally to the tailings, in which Se exhibits its highest relative concentration (i.e. CF = 1.53). Chalcopyrite may contain very minor Se in solid solution.

The highest Cu concentration factor is recorded in the Cu-concentrate (i.e. 5.03), whereas that of Zn is recorded in the Zn-concentrate, (i.e. 19.3). This reflects the enrichment ratios of 16.7 for Cu and 25.6 for Zn, being inversely related to the recovery and economic value of these two valuable metals.

TABLE 10

CONCENTRATION FACTORS FOR 15 ELEMENTS IN THE 17 HBMS MILL PRODUCTS.

PRODUCT.	Au	Ag	Cu	Zn	Cd	Fe	S	Pb	Co	As	Te	Hg	Se	Sb	Sn
1. Feed.	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00
2. Cu-Scav.Co.	2.13	1.70	2.13	1.08	0.95	0.47	0.72	0.95	0.73	0.98	2.85	1.28	0.86	1.15	0.73
3. Cu-Roug.Co.	2.36	2.63	3.05	1.15	0.97	0.49	0.80	1.03	0.73	0.60	2.36	1.23	0.93	1.17	0.66
4. Cu-1st Cl.Fd.	2.36	2.54	2.87	1.16	1.02	0.49	0.80	1.02	0.74	0.61	2.42	1.23	0.92	1.14	0.65
5. Cu-1st Cl.Co.	2.94	3.42	4.36	1.20	1.02	0.41	0.74	1.03	0.57	0.52	2.78	1.29	0.94	1.32	1.05
6. Cu-2nd Cl.Co.	3.05	3.77	5.03	1.20	1.01	0.41	0.75	1.06	0.52	0.50	2.67	1.31	0.98	1.41	1.13
7. Zn-Roug.Fd.	0.79	0.60	0.42	1.23	1.16	1.37	1.30	1.23	1.39	1.39	1.04	1.23	1.25	1.17	1.25
8. Zn-Roug.Co.	1.49	1.73	0.52	8.66	6.75	0.45	1.21	2.37	0.51	1.25	0.48	6.24	0.66	1.39	1.17
9. Zn-1st Cl.Co.	1.76	2.23	0.60	12.79	9.95	0.43	1.47	3.12	0.47	1.15	0.41	8.86	0.71	1.67	0.82
10. Zn-2nd Cl.Co.	2.39	2.97	0.83	19.33	14.94	0.53	2.03	4.32	0.60	1.45	0.38	12.95	0.96	2.34	0.27
11. Cu-Roug.Tl.	0.76	0.58	0.43	1.19	1.19	1.40	1.32	1.23	1.43	1.38	1.07	1.20	1.29	1.17	1.20
12. Cu-Scav.Tl.	0.76	0.58	0.43	1.19	1.19	1.40	1.32	1.23	1.43	1.38	1.07	1.20	1.29	1.17	1.20
13. Cu-1st Cl.Tl.	2.30	2.02	1.32	1.57	1.36	0.83	1.22	1.42	1.34	1.02	2.82	1.63	1.25	1.29	0.20
14. Cu-2nd Cl.Tl.	2.56	1.88	1.31	1.29	1.09	0.50	0.81	0.94	0.92	0.72	3.28	1.32	0.77	0.98	0.69
15. Zn-1st Cl.Tl.	1.27	1.46	0.49	1.98	1.71	0.57	0.93	1.47	0.66	1.58	0.72	2.38	0.63	1.06	2.51
16. Zn-2nd Cl.Tl.	1.33	1.78	0.51	4.63	3.76	0.45	1.01	2.11	0.49	1.12	0.82	4.36	0.53	1.11	2.74
17. Zn-Roug.Tl.	0.88	0.62	0.35	0.78	0.88	1.72	1.56	1.37	1.75	1.62	1.31	0.98	1.53	1.38	1.53



LEGEND.

PRODUCT.

1. Feed.
2. Cu Scavenger Concentrate.
3. Cu Rougher Concentrate.
4. Cu 1st Cleaner Feed.
5. Cu 1st Cleaner Concentrate.
6. Cu 2nd Cleaner Concentrate.

7. Zn Rougher Feed.
8. Zn Rougher Concentrate.
9. Zn 1st Cleaner Concentrate.
10. Zn 2nd Cleaner Concentrate.

11. Cu Rougher Tailings.
12. Cu Scavenger Tailings.
13. Cu 1st Cleaner Tailings.
14. Cu 2nd Cleaner Tailings.

15. Zn 1st Cleaner Tailings.
16. Zn 2nd Cleaner Tailings.
17. Zn Rougher Tailings.

FIGURE 3.
CONCENTRATION FACTOR CURVES FOR THE 15 ELEMENTS IN THE 17 HBMS MILL PRODUCTS.

Chapter VI

MINERAL QUANTITIES ANALYSIS AND MINERAL BALANCES

This chapter describes the evaluation of the modal quantities of the principal ore minerals (See Table 11) in the feed and various products. Mineral quantities were determined by the image analysis system, using both the major and minor mineral routines (See PETROGRAPHIC AND IMAGE ANALYSIS). This involved measuring the relative areal fraction of each mineral or mineral group (e.g. gangue) within the polished section. The data are presented in the equivalent units of volume %.

The major mineral routine determined the mineral quantities of gangue, pyrite, pyrrhotite, chalcopyrite, sphalerite, arsenopyrite, freibergite and galena. Minor cubanite and slag-derived covellite, chalcocite, bornite and Iss contribute to the chalcopyrite mineral quantities, whereas minor marcasite and magnetite contribute to the pyrite mineral quantities. The minor mineral routine re-analyzed the mineral quantities of freibergite and galena with greater precision and accuracy, and additionally analyzed those of pyrargyrite, hessite, electrum and gudmundite. Because of the extremely trace abundances and relatively few measurements possible for grains of these minor minerals in the various products (except for galena), the raw mineral quantities contain large uncertainties. Determination of the modal quantities of the ultra-trace minerals cassiterite, bournonite, rucklidgeite, clausthalite and guanajuatite were not undertaken.

The raw modal analyses for the 20 mill product samples are presented in Table 12, and are condensed for the 17 mill products in Table 13. The most notable feature is that pyrargyrite, hessite and electrum were undetected in many of the mill products. Furthermore, of the trace minerals observed in only a few samples, the value of area % given often constitutes that of only one or two grains i.e. one grain of electrum in the tailings. Therefore, the raw values for the mineral quantities of the minor minerals only provide imprecise data on the relative magnitude of the abundances of these minerals. These data are of interest (1) in estimating the general distribution throughout the circuit of the various minerals of interest, and (2) in determining their absolute abundances for the purpose of metal accounting. As the composition of each of these minerals is either fixed stoichiometrically or known with reasonable confidence, the determination of absolute abundances of these minerals in the various products may permit mineral-metal accounting to be done.

TABLE 11

LIST OF ORE MINERALS OBSERVED IN THE HBMS MILL PRODUCTS.

NAME	FORMULA	ABUNDANCE
COMMON SULFIDES.		
Pyrite	FeS_2	Ubiquitous, Major/Minor.
Chalcopyrite	$CuFeS_2$	Ubiquitous, Major/Minor.
Sphalerite	ZnS	Ubiquitous, Major/Minor.
Pyrrhotite	$Fe_{(1-x)}S$	Common, Major/Trace.
Marcasite	FeS_2	Common, Major/Trace.
Galena	PbS	Common, Major/Trace.
Arsenopyrite	$FeAsS$	Common, Major/Trace.
Cubanite	$CuFe_2S_3$	Occassional, Trace.
OXIDES.		
Magnetite	Fe_3O_4	Occassional, Major/Minor.
Cassiterite	SnO_2	Rare, Trace.
ALLOYS.		
Mercurian Electrum	Ag,Au,Hg	Common, Trace.
ANTIMONIAL SULFOSALTS.		
Freibergite	$(Cu,Ag)_{10}(Fe,Zn)_2(Sb,As)_4S_{13}$	Occassional, Minor/Trace.
Gudmundite	$FeSbS$	Occassional, Trace.
Pyrargyrite	Ag_3SbS_3	Rare, Trace.
Bournonite	$PbCuSbS_3$	Rare, Trace.
TELLURIDES.		
Hessite	Ag_2Te	Rare, Trace.
Rucklidgeite	$(Bi,Pb)_3Te_4$	Rare, Trace.
SELENIDES.		
Clausthalite	$PbSe$	Occassional, Minor/Trace.
Guanajuatite	Bi_2Se_3	Rare, Trace.

Presented in order of decreasing abundance of the mineral groups, and of the minerals in each group. Slag derived sulfides are not included.

MATERIAL BALANCES OF THE MINERAL QUANTITIES

The raw mineral quantities were adjusted by material balance calculations in order to simulate their behaviour through the circuit under the steady state conditions. The adjusted mineral quantities are given in Table 14, while the calculated recoveries are given in Table 15. It is apparent by comparing Tables 13 and 14 that the adjustments to the raw data are relatively minor for all the minerals measured by the major mineral routine, as well as galena, whereas those for freibergite and gudmundite are high. Electrum and pyrargyrite were observed in only four, and hessite in three of the twelve products examined by the minor mineral routine. The scarcity of observations of these minerals prohibited material balance simulation of their behaviour through the circuit. Also shown in Table 14, under the column entitled 'Total' is the sum of these adjusted mineral quantities for each product. It is apparent that there is a progressively worsening cross-balancing deficit in the Cu-circuit concentrates (i.e. 99.92 to 94.60). The cause of these systematic errors is not apparent, while the magnitude of the errors is tolerable (i.e. approx. 5 %).

Calculated Cu and Zn assays derived from the adjusted mineral quantities are 16.8 % Cu in the Cu-2nd cleaner concentrate and 40.2 % Zn in the Zn-2nd cleaner concentrate. These compare satisfactorily with the adjusted chemical assays of 17.4 % Cu and 46.2 % Zn in the respective concentrates, especially in view of the small amount of material that was examined by image analysis compared to chemical analysis.

The mineral recoveries given in Table 15 are plotted as histograms in Figure 4, and graphically describe the behaviour of each of the minerals of interest through the circuit. Those minerals exhibiting similar behaviour are grouped as follows.

Group 1 consists of gangue, pyrite and gudmundite, all of which behave as essentially inert minerals with recoveries greater than 90 % in the tailings. Whereas gangue exhibits less than 6 % recovery in all overflow/concentrate streams, both pyrite and gudmundite show significant recovery (i.e. 25.8 and 14.1 %, respectively) in the Cu-rougher concentrate. These are largely rejected during Cu-cleaning with final recoveries in Cu-concentrate of 7.5 % each. The high pyrite content of the Cu-rougher concentrate principally reflects the recovery of unliberated pyrite occurring in particles as clusters of pyrite grains enveloped in a veneer of chalcopyrite (See Plate 1A and B). In contrast to in the Cu-circuit, gangue, pyrite and (probably) gudmundite exhibit low recoveries in the Zn-rougher concentrate, with final recoveries in the Zn-concentrate of less than 3 % each.

Group 2 consists of arsenopyrite, pyrrhotite and galena, all of which exhibit high recoveries (i.e. 65 to 90 %) in the tailings, but with minor recoveries (i.e. 5 to 20 %) in the Cu- and Zn-concentrates. While the recovery of galena is only 22.1 % in the Cu-rougher concentrate, that of arsenopyrite and pyrrhotite is approximately 30 % each. However, while arsenopyrite and pyrrhotite are largely rejected during Cu-cleaning, the bulk of the galena in

TABLE 12

RAW MINERAL QUANTITIES (BY VOL %) OF THE MINERALS OF INTEREST IN THE 20 HBMS MILL PRODUCT SAMPLES.

SAMPLE.	MAJOR MINERAL ROUTINE							MINOR MINERAL ROUTINE					
	Gang.	Py.	Po.	Sphal.	Cpy.	Aspy.	Gal.	Gal.	Freib.	Pyrarg.	Hess.	Elect.	Gudmun.
1. Feed.	74.3	21.4	0.01	2.03	1.89	0.254	0.081	0.027	0.00074	-------	-------	-------	0.0012
8. Cu-Scav.Co.	62.0	26.9	0.23	7.05	3.01	0.576	0.175	0.026	0.01060	0.00033	-------	-------	0.0006
10. Cu-Scav.Co.	63.3	25.4	0.20	8.83	1.77	0.324	0.058	na.	na.	na.	na.	na.	na.
2. Cu-Roug.Co.	30.6	38.6	0.24	26.4	3.54	0.515	0.122	na.	na.	na.	na.	na.	na.
4. Cu-Roug.Co.	42.2	32.6	0.39	20.3	4.04	0.303	0.115	0.175	0.00626	0.00020	-------	0.00011	0.0007
6. Cu-Roug.Co.	33.2	39.0	0.78	22.3	4.37	0.364	0.086	0.064	0.01319	0.00024	-------	0.00507	0.0011
7. Cu-1st Cl.Fd.	30.9	40.8	0.44	22.7	4.50	0.441	0.088	0.049	-------	-------	-------	-------	0.0009
12. Cu-1st Cl.Co.	16.8	30.8	0.23	45.2	6.46	0.241	0.109	0.133	0.01851	-------	-------	-------	0.0001
14. Cu-2nd Cl.Co.	19.6	23.0	0.61	48.8	7.22	0.514	0.131	0.114	0.05041	-------	0.00005	-------	0.0001
15. Zn-Roug.Fd.	81.0	16.5	0.16	0.25	1.88	0.192	0.035	na.	na	na.	na.	na.	na.
16. Zn-Roug.Co.	18.1	32.7	0.93	2.95	44.4	0.582	0.231	na.	na.	na.	na.	na.	na.
18. Zn-1st Cl.Co.	10.5	26.7	0.82	2.96	57.5	0.971	0.523	0.281	0.00079	0.00018	-------	-------	0.0020
20. Zn-2nd Cl.Co.	6.71	18.4	1.77	1.80	70.2	0.667	0.432	0.282	0.00056	-------	-------	-------	0.0006
3. Cu-Roug.Tl.	82.2	16.0	0.10	0.27	1.03	0.305	0.037	na.	na.	na.	na.	na.	na.
5. Cu-Scav.Tl.	84.5	13.4	0.20	0.42	1.30	0.131	0.020	0.048	0.00506	-------	0.00008	0.00029	------
9. Cu-1st Cl.Tl.	49.0	38.9	0.47	6.35	4.67	0.608	0.081	na.	na.	na.	na.	na.	na.
11. Cu-2nd Cl.Tl.	32.1	50.8	0.37	9.94	5.85	0.791	0.132	0.068	0.00088	-------	0.00007	0.00015	------
19. Zn-1st Cl.Tl.	46.9	40.8	0.74	2.02	8.80	0.512	0.162	na.	na.	na.	na.	na.	na.
21. Zn-2nd Cl.Tl.	30.9	41.9	0.47	1.51	23.9	0.903	0.390	0.199	0.00068	-------	-------	-------	0.0021
17. Zn-Roug.Tl.	86.0	12.7	0.26	0.40	0.44	0.160	0.099	0.021	0.00012	0.00025	-------	0.00001	0.0003

Note : 'na.' denotes no analysis, '-------' denotes not detected.

TABLE 13

RAW MINERAL QUANTITIES (BY VOL %) OF THE MINERALS OF INTEREST IN THE 17 HBMS MILL PRODUCTS.

PRODUCT.	MAJOR MINERAL ROUTINE							MINOR MINERAL ROUTINE					
	Gang.	Py.	Po.	Cpy.	Sphal.	Aspy.	Gal.	Gal.	Freib.	Pyrarg.	Hess.	Elect.	Gudmun.
1. Feed.	74.3	21.4	0.01	2.03	1.89	0.254	0.081	0.027	0.00074	-------	-------	-------	0.0012
2. Cu-Scav.Co.	62.7	26.2	0.22	7.94	2.39	0.450	0.117	0.026	0.01060	0.00033	-------	-------	0.0006
3. Cu-Roug.Co.	35.3	36.7	0.47	23.0	3.98	0.394	0.108	0.119	0.00973	0.00022	-------	0.00259	0.0009
4. Cu-1st Cl.Fd.	30.9	40.8	0.44	22.7	4.50	0.441	0.088	0.049	-------	-------	-------	-------	0.0009
5. Cu-1st Cl.Co.	16.8	30.8	0.23	45.2	6.46	0.241	0.109	0.133	0.01851	-------	-------	-------	0.0001
6. Cu-2nd Cl.Co.	19.6	23.0	0.61	48.8	7.22	0.514	0.131	0.114	0.05041	-------	0.00005	-------	0.0001
7. Zn-Roug.Fd.	81.0	16.5	0.16	0.25	1.88	0.192	0.035	na.	na.	na.	na.	na.	na.
8. Zn-Roug.Co.	18.1	32.7	0.93	2.95	44.4	0.582	0.231	na.	na.	na.	na.	na.	na.
9. Zn-1st Cl.Co.	10.5	26.7	0.82	2.96	57.5	0.971	0.523	0.281	0.00079	0.00018	-------	-------	0.0020
10. Zn-2nd Cl.Co.	6.71	18.4	1.77	1.80	70.2	0.667	0.432	0.282	0.00056	-------	-------	-------	0.0006
11. Cu-Roug.Tl.	82.2	16.0	0.10	0.27	1.03	0.305	0.037	na.	na.	na.	na.	na.	na.
12. Cu-Scav.Tl.	84.5	13.4	0.20	0.42	1.30	0.131	0.020	0.048	0.00506	-------	0.00008	0.00029	------
13. Cu-1st Cl.Tl.	49.0	38.9	0.47	6.35	4.67	0.608	0.081	na.	na.	na.	na.	na.	na.
14. Cu-2nd Cl.Tl.	32.1	50.8	0.37	9.94	5.85	0.791	0.132	0.068	0.00088	-------	0.00007	0.00015	------
15. Zn-1st Cl.Tl.	46.9	40.8	0.74	2.02	8.80	0.512	0.162	na.	na.	na.	na.	na.	na.
16. Zn-2nd Cl.Tl.	30.9	41.9	0.47	1.51	23.9	0.903	0.390	0.199	0.00068	-------	-------	-------	0.0021
17. Zn-Roug.Tl.	86.0	12.7	0.26	0.40	0.44	0.160	0.099	0.021	0.00012	0.00025	-------	0.00001	0.0003

Note : 'na.' denotes no analysis, '-------' denotes not detected.

TABLE 14

MATERIAL BALANCE ADJUSTED MINERAL QUANTITIES OF THE MINERALS OF INTEREST IN THE 17 HBMS MILL PRODUCTS.

PRODUCT.	MAJOR MINERAL ROUTINE.							MINOR MINERAL ROUTINE.			
	Gang.	Py.	Po.	Cpy.	Sphal.	Aspy.	Gal.	Gal.	Freib.	Gudmun.	Total.
1. Feed.	78.79	15.35	0.15	2.41	1.50	0.18	0.034	0.032	0.0010	0.0004	98.41
2. Cu-Scav.Co.	62.70	26.21	0.22	7.91	2.39	0.45	0.117	0.026	0.0107	0.0006	99.92
3. Cu-Roug.Co.	34.15	35.03	0.42	22.71	4.81	0.43	0.096	0.062	0.0063	0.0005	97.62
4. Cu-1st Cl.Fd.	33.85	36.38	0.42	21.69	4.89	0.45	0.099	0.063	0.0059	0.0004	97.80
5. Cu-1st Cl.Co.	19.09	29.48	0.30	39.81	6.22	0.31	0.121	0.108	0.0125	0.0005	95.33
6. Cu-2nd Cl.Co.	16.71	24.73	0.29	46.20	6.30	0.24	0.120	0.117	0.0150	0.0006	94.60
7. Zn-Roug.Fd.	81.46	15.16	0.15	0.29	1.34	0.18	0.031	nc.	nc.	nc.	98.61
8. Zn-Roug.Co.	19.71	28.59	0.95	2.19	46.78	0.64	0.289	nc.	nc.	nc.	99.15
9. Zn-1st Cl.Co.	10.52	24.05	0.97	2.12	60.93	0.75	0.372	0.278	0.0007	0.0009	99.62
10. Zn-2nd Cl.Co.	6.56	20.36	1.06	2.23	68.17	0.71	0.366	0.292	0.0007	0.0007	99.38
11. Cu-Roug.Tl.	81.77	14.92	0.14	0.29	1.27	0.18	0.030	0.028	0.0003	0.0004	98.60
12. Cu-Scav.Tl.	81.81	14.90	0.14	0.27	1.27	0.18	0.030	0.028	0.0003	0.0004	98.60
13. Cu-1st Cl.Tl.	46.28	42.19	0.52	6.41	3.77	0.56	0.080	0.024	0.0003	0.0004	99.75
14. Cu-2nd Cl.Tl.	30.38	51.97	0.34	10.00	5.83	0.66	0.127	0.067	0.0009	0.0003	99.25
15. Zn-1st Cl.Tl.	43.01	43.18	0.74	2.13	8.88	0.50	0.153	nc.	nc.	nc.	98.59
16. Zn-2nd Cl.Tl.	30.87	42.97	0.46	1.55	23.80	0.93	0.404	0.200	0.0007	0.0023	100.78
17. Zn-Roug.Tl.	83.07	14.81	0.13	0.24	0.15	0.17	0.024	0.024	0.0003	0.0003	98.59

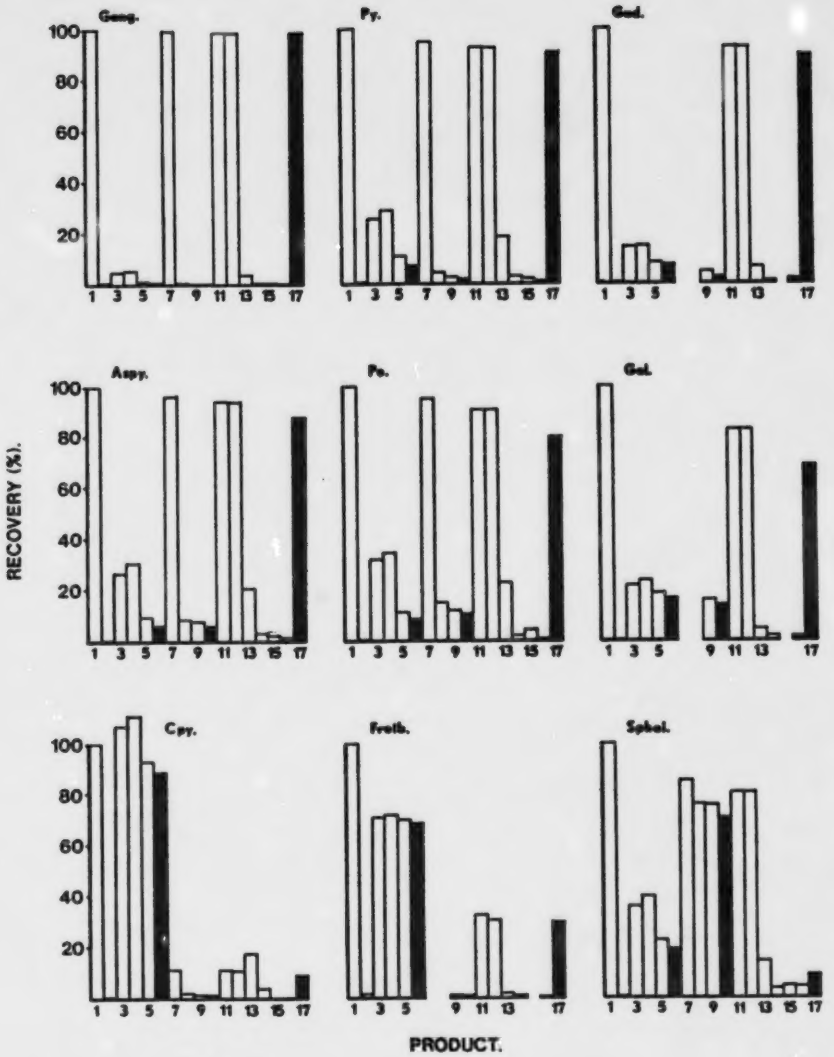
Note : 'nc.' denotes not material balance calculated by MATBAL.

TABLE 15

MATERIAL BALANCE CALCULATED RECOVERIES OF THE MINERALS OF INTEREST IN THE 17 HBMS MILL PRODUCTS.

PRODUCT.	MAJOR MINERAL ROUTINE.							MINOR MINERAL ROUTINE.		
	Gang.	Py.	Po.	Cpy.	Sphal.	Aspy.	Gal.	Gal.	Freib.	Gudmun.
1. Feed.	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0
2. Cu-Scav.Co.	0.14	0.30	0.26	0.57	0.28	0.44	0.61	0.14	1.90	0.29
3. Cu-Roug.Co.	4.91	25.84	32.26	106.9	36.27	27.08	32.43	22.1	71.1	14.1
4. Cu-1st Cl.Fd.	5.29	29.16	34.53	111.0	40.09	30.70	36.13	24.2	72.0	14.9
5. Cu-1st Cl.Co.	1.36	10.81	11.25	93.22	23.13	9.83	20.28	19.1	70.0	8.31
6. Cu-2nd Cl.Co.	0.99	7.48	8.98	89.15	19.50	6.21	16.58	17.0	69.1	7.50
7. Zn-Roug.Fd.	99.50	95.00	95.44	11.64	85.72	96.23	87.43	nc.	nc.	nc.
8. Zn-Roug.Co.	0.61	4.56	15.67	2.24	76.25	8.71	20.99	nc.	nc.	nc.
9. Zn-1st Cl.Co.	0.25	2.93	12.21	1.65	75.86	7.84	20.64	16.2	1.24	4.82
10. Zn-2nd Cl.Co.	0.13	2.08	11.26	1.45	71.03	6.24	16.99	14.3	1.03	2.91
11. Cu-Roug.Tl.	99.15	92.81	91.28	11.42	80.78	94.23	84.03	83.1	32.8	92.8
12. Cu-Scav.Tl.	99.02	92.52	91.03	10.85	80.50	93.79	83.43	83.0	30.9	92.5
13. Cu-1st Cl.Tl.	3.93	18.35	23.28	17.79	16.77	20.88	15.85	5.10	1.99	6.55
14. Cu-2nd Cl.Tl.	0.38	3.33	2.27	4.07	3.82	3.62	3.70	2.06	0.88	0.82
15. Zn-1st Cl.Tl.	0.48	2.48	4.42	0.79	5.22	2.46	4.00	nc.	nc.	nc.
16. Zn-2nd Cl.Tl.	0.12	0.85	0.95	0.20	4.83	1.60	3.67	1.93	0.22	1.95
17. Zn-Roug.Tl.	98.88	90.44	79.77	9.40	9.47	87.55	66.44	68.7	29.9	89.6

Note : 'nc.' denotes not material balance calculated by MATBAL.



LEGEND.

PRODUCT.

1. Feed.
2. Cu Scavenger Concentrate.
3. Cu Rougher Concentrate.
4. Cu 1st Cleaner Feed.
5. Cu 1st Cleaner Concentrate.
6. Cu 2nd Cleaner Concentrate.

7. Zn Rougher Feed.
8. Zn Rougher Concentrate.
9. Zn 1st Cleaner Concentrate.
10. Zn 2nd Cleaner Concentrate.
11. Cu Rougher Tailings.
12. Cu Scavenger Tailings.
13. Cu 1st Cleaner Tailings.
14. Cu 2nd Cleaner Tailings.

15. Zn 1st Cleaner Tailings.
16. Zn 2nd Cleaner Tailings.
17. Zn Rougher Tailings.

FIGURE 4.
THE DISTRIBUTION OF THE RECOVERIES CALCULATED BY MATBAL FOR THE MINERALS OF INTEREST IN THE 17 HBMS MILL PRODUCTS.

the Cu-rougher concentrate reports to the Cu-concentrate (i.e. 17.0 % reduced from 22.1 %). These minerals are interpreted to be essentially inert, and their behaviour is partly modified by the host minerals of the composite particles in which they occur. They show comparable recoveries in the Cu- and Zn-concentrates (i.e. 6.2 % each for arsenopyrite; 9.0 and 11.3 %, respectively for pyrrhotite; and 17.0 and 14.3 %, respectively for galena), suggesting comparable distribution amongst these products, of host composite particles containing chalcopyrite and sphalerite.

Group 3 consists of chalcopyrite and freibergite, both of which float readily into the Cu-circuit concentrates and show high recoveries (i.e. 89.2 and 69.1 %, respectively) in the Cu-concentrate. While chalcopyrite exhibits moderate rejection and hence circulating loads during Cu-cleaning, the freibergite recovered in the Cu-rougher concentrate is essentially fully recovered during Cu-cleaning. Both minerals exhibit low recoveries (i.e. 1 to 2 %) in the Zn-concentrate. Only 9.4 % of the chalcopyrite, but 29.9 % of the freibergite are lost to the tailings, and principally reflect losses as unliberated grains of these minerals, most probably as middling and sub-middling grains.

Group 4 consists of sphalerite only, which shows high recovery in the Zn-concentrate (i.e. 71.0 %), moderate recovery in the Cu-concentrate (i.e. 19.5 %), and minor recovery in the tailings (i.e. 9.5 %). Sphalerite is clearly the only mineral activated by the addition of $CuSO_4$, and hence preferentially floats into the Zn-concentrate. However, the recovery of 19.5 % of the sphalerite in the Cu-concentrate is in part due to the recovery of free grains of sphalerite, as well as unliberated grains of sphalerite. Sphalerite losses to the tailings are principally as unliberated grains locked with pyrite or gangue.

It is particularly significant that while the recovery of Cu in the Cu-concentrate is 77.5 %, that of chalcopyrite is 89.2 %. Similarly, while the recovery of Zn in the Zn-concentrate is 40.2 %, that of sphalerite is 71.0 %. This clearly demonstrates that the losses of potentially recoverable Cu and Zn, as chalcopyrite and sphalerite, respectively, are substantially less than indicated by Cu and Zn assays. The latter include the fraction of unrecoverable, re-cycled Cu and Zn in slag silicates. Nonetheless, the recovery of 89.2 % of the chalcopyrite and particularly 71.0 % of the sphalerite in the respective HBMS concentrates, are lower than the corresponding 96.2 % and 84.9 % recoveries in the respective Trout Lake concentrates (See Healy & Petruk 1989).

CONCENTRATION FACTORS

Two sets of concentration factors (See CHEMICAL ANALYSIS) were calculated for the material balance adjusted mineral quantities of the minerals. The first set were calculated for the 17 mill products, but because the mineral quantities of freibergite and gudmundite were not determined for all the products, freibergite and gudmundite were omitted from the dataset (See Table 16A). The second set of concentration factors were calculated for the 14 products in which mineral quantities for all these minerals were determined, and hence includes freibergite and gudmundite (See Table 16B). Because concentration factors are comparative values, the two sets are inequivalent and not amenable to integration. The following discussion deals with the second set of concentration factors, so that all minerals can be compared, though references to the data in Table 16A are made.

Concentration factors for 9 minerals in the 14 mill products, calculated by 'CONFAC', are given in Table 16B and plotted as discontinuous curves in Figure 5. The results give four mineral groups that contrast with those indicated by the plot of recoveries in Figure 4. Because the concentration of each of the minerals is compared against one another, the relative concentration or depletion of the minerals are more clearly revealed in Figure 5.

Ideally the chalcopyrite curves show the behaviour of minerals that are concentrated in the Cu-concentrate; sphalerite curves the behaviour of minerals that are concentrated in the Zn-concentrate; and gangue the behaviour of minerals that are rejected to the tailings. Comparison of the curves for other minerals to these indicates the tendency for other minerals to concentrate in the Cu-concentrate, Zn-concentrate and tailings. As 98.9 % of the gangue reports to the tailings it essentially represents this tendency, whereas the loss of 9.40 % chalcopyrite to the tailings, and 19.5 % and 9.47 % of the sphalerite to the Cu-concentrate and tailings, respectively, detract from the tendency. Nonetheless, sphalerite and chalcopyrite show the highest relative concentration (i.e. CF's = 82.7 and 16.7, respectively) in the Zn- and Cu-concentrates, respectively, and reflect mineral enrichment ratios of 45.4 and 19.2, respectively.

Because of the overwhelming rejection of gangue to the tailings (i.e. 98.9 %), the remaining minerals have higher modal abundances in most of the products, particularly in the Cu- and Zn-concentrates, relative to the feed. Consequently, these minerals exhibit relative concentration in both the Cu- and Zn-concentrates. However, because of self-dilution and the proportionally large abundances of gangue (despite constituting very minor recoveries), the magnitude of relative concentration is small. The lower mineral quantities of gangue, pyrite and chalcopyrite in the Zn-concentrate (i.e. 6.56, 20.4 and 2.23 %, respectively) versus those of gangue, pyrite and sphalerite in the Cu-concentrate (i.e. 16.7, 24.7 and 6.30 %, respectively) account for the much higher grades, enrichment ratios and concentration factors for Zn and sphalerite in the Zn-concentrate, than for chalcopyrite in the Cu-concentrate.

The Cu-Circuit : The Cu-rougher produces relative concentration of all minerals except gangue and gudmundite, while the Cu-scavenger produces significant relative depletion in gangue and galena. Gangue and arsenopyrite are the only minerals that exhibit their highest concentration factor (in Cu-circuit concentrates) in the Cu-scavenger concentrate. This unit is expected to preferentially recover both the coarse-grained ('over-sized') free and sub-free chalcopyrite and middling particles containing chalcopyrite. Petrographic examination of the Cu-scavenger concentrate shows that negligible over-sized free or sub-free chalcopyrite is recovered in an essentially fine-grained product. Middling particles of pyrite locked with chalcopyrite and/or sphalerite are relatively common, though sphalerite and pyrite are largely free. These observations contrast with those on the Trout Lake Cu-scavenger concentrate, which is extremely coarse-grained with abundant over-sized chalcopyrite, middling sphalerite and chalcopyrite (Healy & Petruk 1989). Because of the poorer grind and liberation of the HBMS circuit, the Cu-rougher is 'pulling hard', such that essentially all marginally floatable particles are already recovered prior to the Cu-scavenger. This also contributes to the extremely small stream flow rate (i.e. 0.174). The preferential recovery of gangue and arsenopyrite in this, rather than other Cu-circuit concentrates, reflects the generally inert nature of the constituent particles recovered.

Gangue is the only mineral (group) that exhibits progressive relative depletion in the Cu-circuit, and strong relative depletion (i.e. CF = 0.19) in the Cu-2nd cleaner concentrate. The mineral quantities of gangue drop from 78.9 % in the feed to 16.7 % in the Cu-concentrate, and this alone elevates the proportions of all other minerals relative to the feed. While the enrichment ratio for chalcopyrite in the Cu-concentrate is only 19.2, the rejection of 99.0 % of the gangue from the Cu-circuit will alone generate an enrichment ratio of 5 for chalcopyrite. Thus, the relative depletion of gangue alone in the Cu-concentrate and accompanying moderate relative concentration of most other minerals, reflects the efficiency of the Cu-circuit rejection of gangue, but also its limited selectivity in sulfide rejection.

The enrichment ratios for pyrite, pyrrhotite, galena, sphalerite, gangue in the Cu-concentrate are 1.61, 1.93, 3.53, 4.20 and 0.21, respectively, and are reflected in the corresponding concentration factors of 1.14, 1.70, 3.21, 3.69 and 0.19, respectively. The grade of sphalerite in the Cu-concentrate is therefore quadruple that of the feed (i.e. 7.5 Vs 1.8 % Zn), and thus also quadruple the nominally acceptable level. The concentration factors for pyrite, pyrrhotite and arsenopyrite show a pattern of moderately increasing relative concentration (i.e. CF's up to 1.70), whereas those of sphalerite and galena exhibit significant (i.e. CF's up to 3.69), and chalcopyrite and freibergite pronounced (i.e. CF's up to 16.7) increasing relative concentration through Cu-cleaning. Only the freibergite curves resemble those of chalcopyrite in the Cu-circuit, whereas only gudmundite, but notably not sphalerite, partly resemble those of gangue in the Cu-circuit.

Pyrite, pyrrhotite, chalcopyrite, sphalerite and arsenopyrite exhibit significant relative concentration (i.e. CF's between 1.5 and 5.0) in the Cu-1st and 2nd cleaner tailings. It is apparent that while the concentration factors for pyrite and sphalerite are similar in the Cu-circuit, pyrite is preferentially rejected over sphalerite. It is noteworthy that the grade of pyrite in the Cu-1st and 2nd cleaner tailings is approximately twice that in the Cu-concentrate, whereas that of sphalerite in the same products is slightly less than in the Cu-concentrate. This suggests (1) possible activation; (2) poor selective flotation; or (3) poor chalcopyrite-sphalerite liberation.

The Zn-Circuit : Sphalerite, galena, pyrrhotite, arsenopyrite, gudmundite, pyrite, chalcopyrite and freibergite (in that order of magnitude) exhibit relative concentration in the Zn-2nd cleaner concentrate, whereas only gangue exhibits relative depletion. The concentration factor for sphalerite in the Zn-concentrate is 82.7, and greatly exceeds that of the next highest concentration factors (i.e. 16.6 for galena, 12.9 for pyrrhotite and 7.18 for arsenopyrite). The high concentration factor for sphalerite reflects the enrichment ratio of 45.5, which is substantially higher than that for Zn (i.e. 25.7). The latter includes the large component of Zn in slag silicates that is principally lost to the tailings. This indicates that the flotation of sphalerite in the Zn-circuit (using activation by $CuSO_4$) generates high selective recovery of the sphalerite that is rejected by the Cu-circuit.

In the Zn-concentrate, both galena and pyrrhotite exhibit concentration factors of > 10, despite recoveries of 17.0 and 11.3 %, respectively, while arsenopyrite, gudmundite and pyrite exhibit concentration factors of between 2 and 4, despite recoveries of less than 7 %. These minerals are expected to be inert to $CuSO_4$ treatment. Thus, to be recovered in the Zn-circuit, rather than the Cu-circuit implies that (1) their flotation behaviour is partly modified by the degree of liberation and locking with sphalerite, and/or (2) regrinding and further conditioning prior to the Zn-circuit has increased the fraction of floatable grains of some of these minerals. Petrographic examination shows that the bulk of the galena, pyrrhotite, arsenopyrite and pyrite in the Zn-concentrate is locked with sphalerite (See Plate 1C, D, E and F).

Because the gangue is largely rejected initially in the Zn-rougher, reporting directly to the tailings, and chalcopyrite and freibergite have already been largely removed in the Cu-circuit, the concentration factors for pyrite, pyrrhotite, arsenopyrite, galena and gudmundite are high in the Zn-cleaning circuit. Nonetheless, it is apparent from the concentration factors of the latter minerals in the Zn-2nd cleaner tailings, that while the Zn-rougher (no corresponding concentration factors) exhibits moderate selectivity in sulfide flotation, the Zn cleaners yield greater rejection. Thus, while the gangue reports directly to the Zn-rougher tailings (CF's < 0.25 in the three Zn-circuit concentrates), there are minor circulating loads of galena, pyrrhotite, arsenopyrite, pyrite and gudmundite, which are largely rejected during Zn-cleaning. From Table 16A it is apparent that gangue is almost totally rejected by the Zn-rougher, pyrite and pyrrhotite by both the Zn-1st and 2nd cleaners, and arsenopyrite and galena by the Zn-2nd cleaners. The Zn-rougher tailings exhibit relative depletion for chalcopyrite, sphalerite and freibergite only; the only three valuable minerals modelled through the circuit. Conversely, at least moderate relative concentration is exhibited by all other minerals in the Zn-rougher tailings. Gangue, pyrite, pyrrhotite, arsenopyrite, galena and gudmundite all exhibit relatively high relative concentration (i.e. CF's > 2.5) in the Zn-rougher tailings.

Petrographic examination shows that the pyrrhotite reporting to the Zn-concentrate occurs in diablastic and granular composite particles with sphalerite, as rods and blebs of pyrrhotite in sphalerite (See Plate 1F), but also as middling grains locked with chalcopyrite which failed to float in the Cu-circuit. Chalcopyrite exhibits progressively increasing relative concentration through Zn-cleaning (i.e. CF's increase from 0.26 to 1.87; See Table 16A), with negligible rejection to the Zn-rougher or cleaner tailings (i.e. CF's are 0.54, 0.47 and 0.38, respectively; See Table 16A). Petrographic examination of the Zn-concentrate indicates four principal forms of chalcopyrite : (1) middling particles of chalcopyrite locked with pyrite and/or pyrrhotite (See Plate 1F, G and H); (2) coarse-grained (or 'oversized'), free and sub-free chalcopyrite grains (See Plate 1F and G); (3) diablastic and granular chalcopyrite locked with sphalerite (See Plate 1F, G and H); and (4) chalcopyrite-disease in sphalerite (See Plate 1H). The coarse-grained, free and sub-free chalcopyrite are not as conspicuous or as significant as at Trout Lake, but nonetheless correspond to possibly 15 % of the chalcopyrite in the Zn-concentrate.

TABLE 16

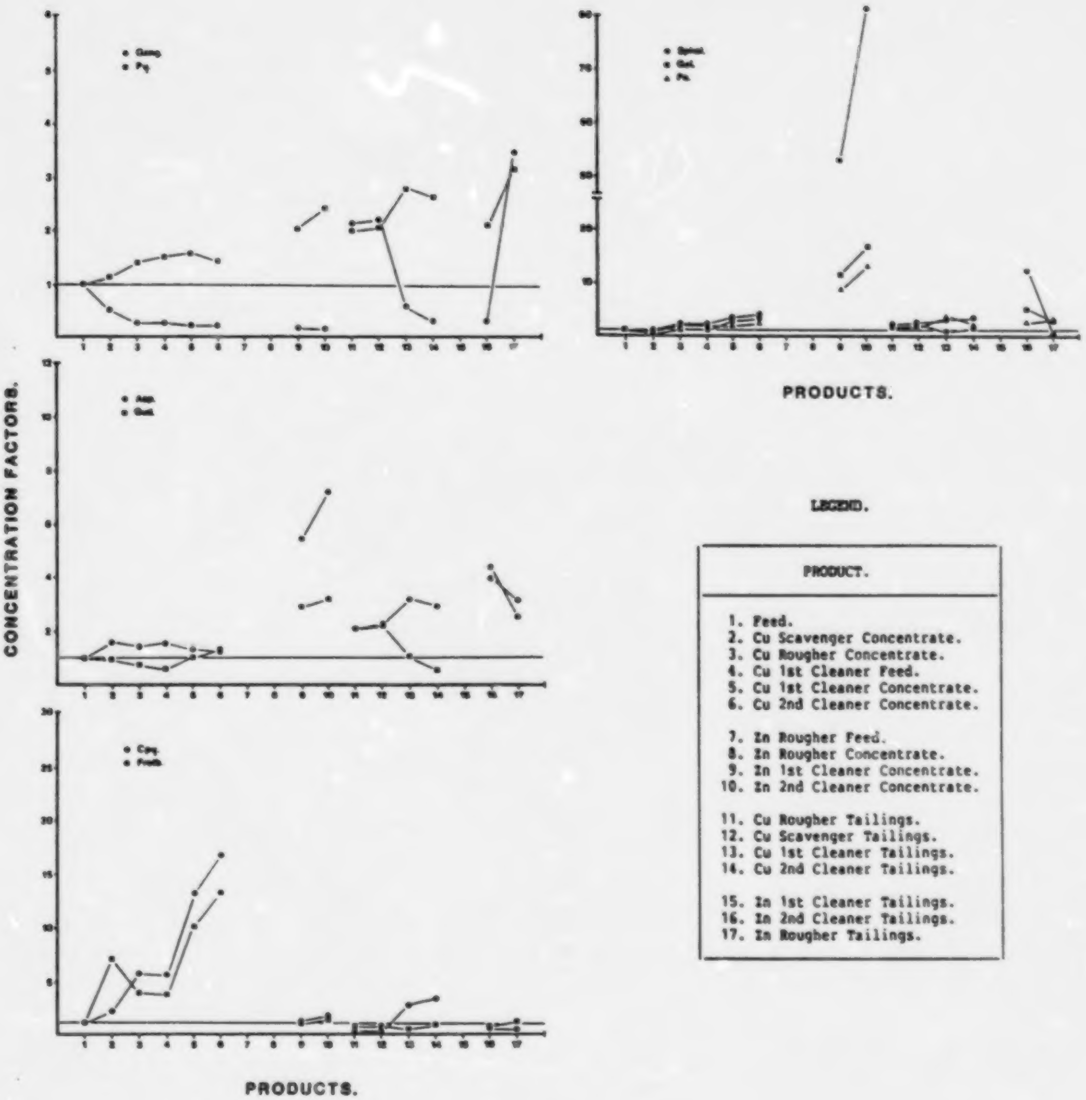
CONCENTRATION FACTORS FOR MINERAL QUANTITIIES IN THE 17 HBMS MILL PRODUCTS.

A.

PRODUCT.	Gang.	Py.	Po.	Cpy.	Sphal.	Aspy.	Gal.
1. Feed	1.00	1.00	1.00	1.00	1.00	1.00	1.00
2. Cu-Scav.Co.	0.47	1.01	0.87	1.93	0.94	1.48	2.04
3. Cu-Roug.Co.	0.27	1.40	1.72	5.75	1.97	1.47	1.73
4. Cu-1st Cl.Fd.	0.26	1.45	1.71	5.46	1.99	1.52	1.78
5. Cu-1st Cl.Co.	0.22	1.73	1.80	16.81	3.74	1.55	3.21
6. Cu-2nd Cl.Co.	0.22	1.65	1.98	19.51	4.31	1.37	3.62
7. Zn-Roug.Fd.	2.07	1.98	2.00	0.26	1.79	2.00	1.82
8. Zn-Roug.Co.	0.22	1.66	5.64	0.80	27.77	3.17	7.87
9. Zn-1st Cl.Co.	0.19	2.19	9.04	1.21	56.81	5.83	15.30
10. Zn-2nd Cl.Co.	0.17	2.72	14.51	1.87	93.29	8.10	22.10
11. Cu-Roug.Tl.	2.10	1.97	1.89	0.26	1.72	2.03	1.79
12. Cu-Scav.Tl.	2.18	2.04	1.96	0.25	1.78	2.10	1.86
13. Cu-1st Cl.Tl.	0.33	1.52	1.92	1.47	1.39	1.72	1.30
14. Cu-2nd Cl.Tl.	0.24	2.11	1.41	2.58	2.43	2.29	2.33
15. Zn-1st Cl.Tl.	0.33	1.72	3.02	0.54	3.62	1.70	2.75
16. Zn-2nd Cl.Tl.	0.29	2.05	2.25	0.47	11.65	3.79	8.72
17. Zn-Roug.Tl.	3.75	3.44	3.09	0.38	0.36	3.36	2.51

B.

PRODUCT.	Gang.	Py.	Po.	Cpy.	Sphal.	Aspy.	Gal.	Freib.	Gudmun.
1. Feed.	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00
2. Cu-Scav.Co.	0.52	1.11	0.95	2.12	1.04	1.62	0.53	6.95	0.97
3 Cu-Roug.Co.	0.26	1.37	1.68	5.64	1.93	1.44	1.17	3.79	0.75
4. Cu-1st.Cl.Fd.	0.27	1.48	1.74	5.57	2.03	1.56	1.23	3.67	0.62
5. Cu-1st.Cl.Co.	0.19	1.54	1.60	13.16	3.32	1.38	2.70	10.01	1.00
6. Cu-2nd.Cl.Co.	0.19	1.41	1.70	16.70	3.69	1.17	3.21	13.17	1.32
9. Zn-1st.Cl.Co.	0.17	2.03	8.40	1.13	52.75	5.41	11.20	0.91	2.92
10. Zn-2nd.Cl.Co.	0.15	2.41	12.87	1.66	82.74	7.18	16.61	1.27	3.19
11. Cu-Roug.Tl.	2.14	2.00	1.92	0.26	1.74	2.06	1.80	0.62	2.06
12. Cu-Scav.Tl.	2.20	2.06	1.98	0.25	1.79	2.12	1.85	0.64	2.12
13. Cu-1st.Cl.Tl.	0.60	2.78	3.51	2.68	2.55	3.15	0.76	0.30	1.01
14. Cu-2nd.Cl.Tl.	0.30	2.64	1.77	3.23	3.03	2.86	1.63	0.70	0.99
16. Zn-2nd.Cl.Tl.	0.30	2.12	2.32	0.49	12.02	3.91	4.73	0.53	4.35
17. Zn-Roug.Tl.	3.46	3.16	2.84	0.35	0.33	3.10	2.46	0.98	2.46



LEGEND.

PRODUCT.

1. Feed.
2. Cu Scavenger Concentrate.
3. Cu Rougher Concentrate.
4. Cu 1st Cleaner Feed.
5. Cu 1st Cleaner Concentrate.
6. Cu 2nd Cleaner Concentrate.

7. Zn Rougher Feed.
8. Zn Rougher Concentrate.
9. Zn 1st Cleaner Concentrate.
10. Zn 2nd Cleaner Concentrate.

11. Cu Rougher Tailings.
12. Cu Scavenger Tailings.
13. Cu 1st Cleaner Tailings.
14. Cu 2nd Cleaner Tailings.

15. Zn 1st Cleaner Tailings.
16. Zn 2nd Cleaner Tailings.
17. Zn Rougher Tailings.

FIGURE 5.
THE CONCENTRATION FACTORS CURVES FOR MINERAL QUANTITIES OF THE MINERALS IN THE 17 HBMS MILL PRODUCTS.

COVELLITE AND CHALCOCITE

Covellite along with the related sulfides chalcocite and digenite are known to react in the pulp, and when present in amounts greater than 0.01 Wt % cause activation of sphalerite (Petruk & Mainwaring 1985). Healy & Petruk (1989) reported that abundant covellite had been identified in the HBMS mill products, and that it is derived from Cu-Fe-sulfide phases entrained in Cu-smelter slag (See Plate 2A, B and H). The latter is introduced into the feed ore by mine dilution from backfilled slag.

Further petrographic examination revealed that some of what was initially identified as covellite is in fact chalcocite, a closely related phase (chemically and optically). Generally, the covellite occurs as overgrowths and rims consisting of radial aggregates on, and fracture-fillings in, the slag sulfide and silicate phases. In contrast, chalcocite typically occurs as spindle-shaped exsolution lamellae in the slag sulfide phases intermediate solid solution (Iss) and bornite, as well as granular masses (See Plate 2A).

Healy & Petruk (1989) conducted a microprobe study of these phases and confirmed the presence of S-rich species of Iss, bornite and bornite-chalcocite solid solution, as well as Cu- and Zn-bearing silicates. The compositions lie within the extensive liquid field of the CuFeS system at the temperatures of Cu-smelting (i.e. 1100°C). High magnification petrographic examination reveals that the phases are not homogeneous and consist of extremely fine-grained intergrowths, reflecting partial sub-solidus breakdown during rapid quenching of the slag. Petrographic examination of slag particles sampled from the HBMS rod mill feed revealed the presence of these phases, confirming that they are derived from the slag. Furthermore, the slag silicates are characterized by porphyritic and spinifex textures; textures that are also exhibited by the silicate phases that are commonly associated with the covellite, chalcocite, Iss, bornite, etc. in the mill products (See Plate 2B and H).

The silicate phases consist of variably-textured fayalite (typically radiating prismatic crystals i.e. spinifex texture, or blocky crystals i.e. porphyritic texture) in a matrix of glass. Qualitative electron microprobe analysis of these phases gave peaks of Fe > Si > Zn > Mg as the principal detectable components in fayalite, and peaks of Si > Fe > Ca > Al > K > S > Na > Zn > Cu as the principal detectable components in the glass. Both phases contain oxygen, which is not detectable by the microprobe. The peak heights of Zn and Cu in these phases, and material balance considerations of Cu and Zn assays, and chalcopyrite and sphalerite mineral quantities indicate that these phases contain average contents of 6.5 % Zn and 1.5 % Cu.

The abundance of covellite and chalcocite in the feed probably exceeds that of 0.01 Wt %, the bulk of which is recovered in the Cu-concentrate. These two phases are possibly causing activation of sphalerite, and contributing significantly to the Zn-losses in the Cu-concentrate. Therefore, the use of slag as backfill in stopes likely to be reactivated (e.g. for removal of crown pillars) and the dilution of the ore by slag should be discouraged, where selective flotation of sphalerite is desirable. However, there is no unequivocal evidence for activation, from either cyanidation tests (N McEachern, pers. comm. 1989) or from this study.

PLATE 1.

(A) Cu-1st CLEANER TAILINGS : Abundant middling and sub-middling particles (labelled A to J) containing chalcopyrite (Cp) + pyrite (Py) ± sphalerite (Sp), typically with chalcopyrite forming interstitially to the pyrite grains. These particles contribute significantly to chalcopyrite losses, and conversely to lower Cu-concentrate grades, and are readily amenable to regrinding. Note several grains of free sphalerite and one grain of freibergite (Fr). Plane Polarised Light (PPL).

(B) Cu-CONCENTRATE : Abundant middling and sub-middling particles (labelled A to F) containing chalcopyrite + pyrite ± sphalerite. Note several grains of free sphalerite, and a coarse-grained free grain of freibergite. Note bulk of pyrite is unliberated with at least a minor veneer of chalcopyrite. PPL.

(C) FEED : Large particle consisting of disseminated pyrite grains enveloped by interstitial sphalerite. PPL.

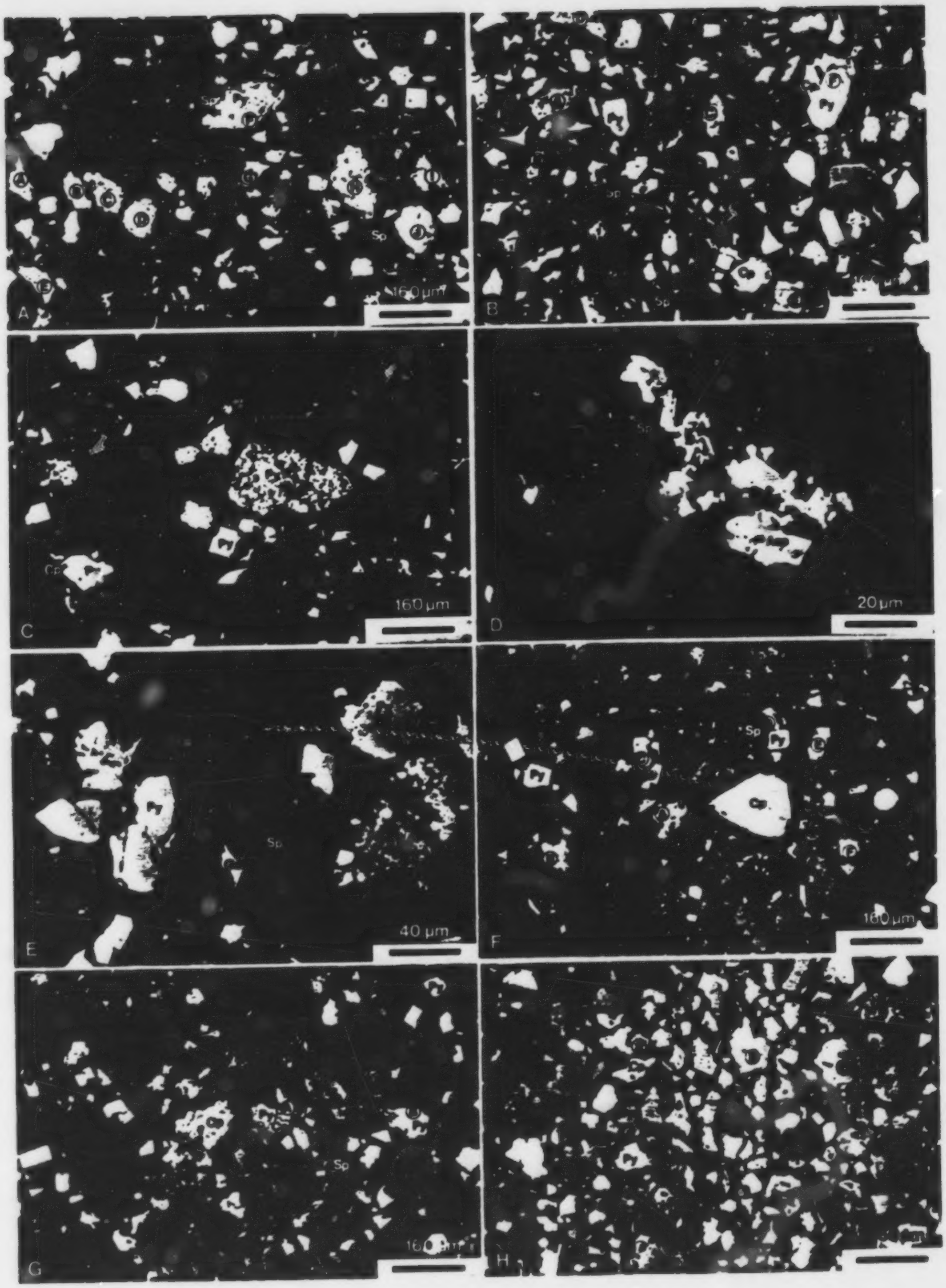
(D) Zn-CONCENTRATE : Particle consisting of clusters of arsenopyrite (Asp) grains in sphalerite. Oil Immersion (OI). PPL.

(E) Zn-CONCENTRATE : Particles of middling and sub-middling sphalerite locked with pyrite or arsenopyrite. Note two grains of galena (Gal) locked with sphalerite. PPL.

(F) Zn-CONCENTRATE : Coarse-grained (i.e. over-sized) free chalcopyrite, and abundant composite particles of chalcopyrite ± pyrite ± sphalerite (i.e. A to F). Note diablastic-textured pyrrhotite (Po) in sphalerite (lower center) and chalcopyrite in sphalerite at upper center. PPL.

(G) Zn-CONCENTRATE : Coarse-grained free chalcopyrite, and composite particles of middling chalcopyrite + pyrite (i.e. A and B), and various particles of sphalerite ± pyrite ± chalcopyrite. Note numerous grains of free pyrite. PPL.

(H) Zn-CONCENTRATE : Diablastic-textured particles of chalcopyrite and sphalerite (i.e. A and B), and numerous particles of sphalerite ± pyrite ± chalcopyrite. Note chalcopyrite-disease in sphalerite (i.e. C), and numerous free grains of pyrite. PPL.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GOLD AND SILVER MINERAL-METAL ACCOUNTING

The scarcity of electrum, pyrargyrite and hessite, and the consequent inability to determine the modal quantities of these minerals in the circuit products prohibits any meaningful mineral-metal balance of Ag and of Au. Furthermore, the high uncertainties in the modal quantities of freibergite and galena, and the lack of microprobe analyses (and hence quantitative determination of the Ag content) of freibergite in the HBMS ore, and the complication due to possible Ag in solid solution in various slag silicate phases, accentuates the problem for Ag particularly.

The low Au assay (i.e. 1.029 ppm), the high Au losses to the tailings (i.e. 46.3 %), the high pyrite and arsenopyrite content of the ore (i.e. 15.4 and 0.18 %, respectively), and the low modal quantities of electrum in all products, suggests that the 'invisible' Au content of the ore is high, compared to the Trout Lake ores (See Healy & Petruk 1990). However, in addition to the losses as 'invisible' Au, significant Au is probably lost to the tailings as electrum locked in pyrite and arsenopyrite, due to the poor grind and low liberation (See Plate 2D and E).

Seven grains of freibergite were observed during petrographic examination of the Cu-concentrate, of which several were relatively coarse-grained (See Plates 1B and 2F). The adjusted modal quantities of freibergite in the feed (i.e. 0.001 vol %) are significantly higher than in the Trout Lake feed (i.e. 0.0002 vol %; See Healy & Petruk 1989). After correcting the modal quantities for specific gravity, the 0.0016 Wt % freibergite accounts for a Ag assay of 3.4 ppm, and thus 27 % of the Ag in the feed. However, this value is based on the assumption that the Ag content of the freibergite is identical to that at Trout Lake (i.e. 21.5 % Ag; See Healy & Petruk 1989). Nonetheless, this suggest that freibergite is a major, and possibly the dominant Ag-carrier in the ore. Based on Ag assays and the recovery of sphalerite in the products, Ag in solid solution in sphalerite can only account for a maximum of 8.5 % of the Ag in the feed. It is inferred that freibergite, chalcopyrite, pyrite, sphalerite, electrum and galena are the principal Ag-carriers in the ore.

Chapter VII

MINERAL LIBERATIONS OF CHALCOPYRITE AND SPHALERITE

In this chapter the liberation and classification of chalcopyrite and sphalerite are evaluated. The liberation (% free) is the relative volume % of each mineral of interest that occurs as free grains; grains having no common boundary with other minerals, and hence not touching or locked with grains of other minerals. The remainder, the unliberated grains occur in composite particles with one or more other minerals. The classification describes the volume % of each grain of the mineral of interest in terms of the total volume of the host particle. Thus, the classification describes the particles in which the mineral of interest occurs in terms of the grains contribution to the particle volume (See PETROGRAPHIC AND IMAGE ANALYSIS).

MATERIAL BALANCES OF THE FREE GRAINS

The raw liberation (% free) for chalcopyrite and sphalerite in each of the 20 mill product samples is presented in Table 1 of Appendix A, and are condensed for the 17 principal mill products in Table 17. The condensed raw liberation data were transformed into a coherent set of data by material balance calculations. The liberations were first weighted in terms of the amount of the mineral(s) present in each product e.g. the mineral quantities of chalcopyrite was multiplied by the liberation of chalcopyrite in that product. These raw modal fractions were transformed by MATBAL into a set of adjusted modal fractions of the respective free mineral, and recoveries for the same were calculated. The adjusted modal fractions were subsequently divided by the respective quantities of the mineral for that product, to give adjusted, but non-coherent liberations. Thus, a set of adjusted liberations is not generated, but is extracted from two sets of coherent data. The extracted liberation data are presented in Table 18.

Comparison of the values in Tables 17 and 18, indicate that in many products the derived adjustments to the raw liberations are relatively large as a proportion of the raw values (e.g. 56.4 to 24.5 % of free chalcopyrite in the Cu-concentrate). This may suggest that the image analysis generated relatively inaccurate modal fractions of free mineral grains in many samples, or that stereological biases (Hill et al. 1986) have heavily influenced the dataset (See PETROGRAPHIC AND IMAGE ANALYSIS). The calculated 24.7 % and 27.9 % free grains of chalcopyrite and sphalerite, respectively include a small, undetermined component of unliberated grains that appear as free grains of the respective minerals. This and other related stereological effects may operate on the present dataset, and an appreciation of their origins is critical to the data interpretation (See Petruk 1978, Hill et al. 1986 and He[illegible] & Petruk 1989). Nonetheless, the effects of stereological biases are

TABLE 17

CONDENSED RAW MINERAL LIBERATIONS IN THE 17 MILL PRODUCTS.

PRODUCT.	CHALCOPYRITE	SPHALERITE
1. Feed.	17.7	15.9
2. Cu-Scav Co.	34.4	63.8
3. Cu-Roug Co.	19.8	32.9
4. Cu-Cl.1 Fd.	20.5	31.0
5. Cu-Cl.1 Co.	33.2	43.6
6. Cu.Cl.2 Co.	56.4	60.2
7. Zn-Roug Fd.	5.04	20.2
8. Zn-Roug Co.	12.1	35.2
9. Zn-Cl.1 Co.	8.91	42.0
10. Zn-Cl.2 Co.	22.0	53.7
11. Cu-Roug Tl.	7.36	16.3
12. Cu-Scav Tl.	5.29	32.2
13. Cu-Cl.1 Tl.	7.37	45.9
14. Cu-Cl.2 Tl.	8.07	42.7
15. Zn-Cl.1 Tl.	6.82	24.8
16. Zn-Cl.2 Tl.	20.0	46.0
17. Zn-Roug Tl.	3.29	5.64

assumed to be minor, and the large adjustments to the raw data, and poor cross-balancing (See below) are attributed to errors in sampling and sample preparation. These effects propagate errors through into the final results generated by MATBAL. For example, as approximately 90 % of the chalcopyrite is recovered in the Cu-concentrate, the raw values of 17.7 % free chalcopyrite in the feed, and 56.4 % free chalcopyrite in the Cu-concentrate are clearly incompatible. Consequently, MATBAL generated corresponding adjusted values of 24.7 % free chalcopyrite in the feed, and 27.1 % in the Cu-concentrate (Compare Tables 17 and 18).

The calculated liberations for chalcopyrite and sphalerite in the circuit feed are 24.7 % and 27.9 %, respectively (See Table 18), and indicate extremely poor liberations, stemming austensibly from the poor grind (i.e. 80 % - 110 um). These liberations compare unfavourably with those of 60.6 % for chalcopyrite and 50.5 % for sphalerite in the Trout Lake feed (Healy & Petruk 1989). The liberation of chalcopyrite in the Cu-concentrate is 27.1 %, indicating the low grade nature of the concentrate due to poor liberation and the consequent need for the circuit to 'pull hard'. Consequently, 97.7 % of

TABLE 18

ADJUSTED MINERAL LIBERATIONS IN THE 17 MILL PRODUCTS.

PRODUCT.	CHALCOPYRITE	SPHALERITE
1. Feed.	24.7	27.9
2. Cu-Scav Co.	29.1	66.7
3. Cu-Roug Co.	23.8	30.4
4. Cu-Cl.1 Fd.	23.2	31.4
5. Cu-Cl.1 Co.	26.2	30.0
6. Cu.Cl.2 Co.	27.1	27.8
7. Zn-Roug Fd.	5.33	27.7
8. Zn-Roug Co.	9.56	30.1
9. Zn-Cl.1 Co.	11.7	31.5
10. Zn-Cl.2 Co.	10.8	30.5
11. Cu-Roug Tl.	6.39	28.0
12. Cu-Scav Tl.	5.19	27.9
13. Cu-Cl.1 Tl.	7.26	33.4
14. Cu-Cl.2 Tl.	8.10	41.1
15. Zn-Cl.1 Tl.	7.27	25.0
16. Zn-Cl.2 Tl.	18.1	47.1
17. Zn-Roug Tl.	4.32	8.44

NOTE : Calculated above values by dividing the material balance adjusted modal fraction of free chalcopyrite by the adjusted modal amounts of chalcopyrite, both of which datasets were calculated independently by MATBAL.

the free chalcopyrite is recovered in the Cu-concentrate, with only 0.63 % reporting to the Zn-concentrate, and 1.64 % to the tailings (See Table 19). These minor losses may correspond to 'over-sized' free chalcopyrite grains (See Plate 1F and G), and/or stereological biases generating apparent free grains derived from unliberated grains in the tailings. Nonetheless, the recovery of 97.7 % of the free grains of chalcopyrite in the Cu-concentrate confirms that the relatively poor Cu-metallurgy of the circuit is not due to any siginficant losses of free chalcopyrite.

The liberation of sphalerite in the Zn-concentrate is 30.5 %, and is comparable to that of the Cu-rougher and Cu-cleaner concentrates. From Table 19 it can be seen that 77.7 % of the free sphalerite is recovered in the Zn-

TABLE 19

RECOVERY OF FREE MINERALS IN THE 17 MILL PRODUCTS.

PRODUCT.	CHALCOPYRITE	SPHALERITE
1. Feed.	100.0	100.0
2. Cu-Scav Co.	0.69	0.67
3. Cu-Roug Co.	103.0	39.5
4. Cu-Cl.1 Fd.	104.3	45.1
5. Cu-Cl.1 Co.	99.1	25.1
6. Cu.Cl.2 Co.	97.7	19.4
7. Zn-Roug Fd.	2.50	85.3
8. Zn-Roug Co.	0.87	82.4
9. Zn-Cl.1 Co.	0.78	85.7
10. Zn-Cl.2 Co.	0.63	77.7
11. Cu-Roug Tl.	2.96	81.2
12. Cu-Scav Tl.	2.27	80.6
13. Cu-Cl.1 Tl.	5.23	20.1
14. Cu-Cl.2 Tl.	1.34	5.63
15. Zn-Cl.1 Tl.	0.23	4.68
16. Zn-Cl.2 Tl.	0.15	8.17
17. Zn-Roug Tl.	1.64	2.88

concentrate, though only 80.6 % actually enters the Zn-circuit (i.e. 96.4 % efficient), with 2.88 % being lost to the tailings. The losses to the tailings are minor, and may correspond to apparent free grains generated by a stereological bias. The principal loss of free sphalerite occurs in the Cu-concentrate (i.e. 19.4 %), where the liberation (i.e. 27.8 %) is comparable to that of sphalerite in the Zn-concentrate (i.e. 30.5 %). This shows that significant losses of free sphalerite occur in the Cu-concentrate, and that there is only a slight preferential recovery of locked over free sphalerite in the Cu-concentrate (See Plate 1B and 2G). These losses to the Cu-concentrate however, represent only 5.4 % of the sphalerite in the feed, but indicate either activation, or poor selective flotation of sphalerite.

In order to assess the intrinsic recovery at each flotation unit, the recovery in the unit concentrate as a percent of the recovery of that unit's feed is calculated for a given mineral. This flotation efficiency expressed in percent, is thus in standardized units that are independent of the behaviour of other minerals, or the preceeding flotation units which have altered the proportion of the minerals entering that unit. Flotation efficiencies were calculated for free chalcopyrite and sphalerite and are given in Table 20. These flotation efficiencies are presented as curves for the Cu- and Zn-

TABLE 20

FLOTATION EFFICIENCIES FOR FREE CHALCOPYRITE AND SPHALERITE.

PRODUCT.	CHALCOPYRITE			SPHALERITE		
	RECOVERY Feed.	RECOVERY Conc.	Effic.	RECOVERY Feed.	RECOVERY Conc.	Effic.
Cu-Scavenger	2.96	0.69	23.3	81.2	0.67	0.83
Cu-Rougher	105.9	103.0	97.3	120.8	39.5	32.7
Cu-1st Cleaner	104.3	99.1	95.0	45.1	25.1	55.7
Cu-2nd Cleaner	99.1	97.7	98.6	25.1	19.4	77.3
Zn-Rougher	2.50	0.87	34.8	85.3	82.4	96.6
Zn-1st Cleaner	1.11	0.78	70.3	90.6	85.7	94.6
Zn-2nd Cleaner	0.78	0.63	80.8	85.7	77.7	90.7

circuit units in Figure 6.

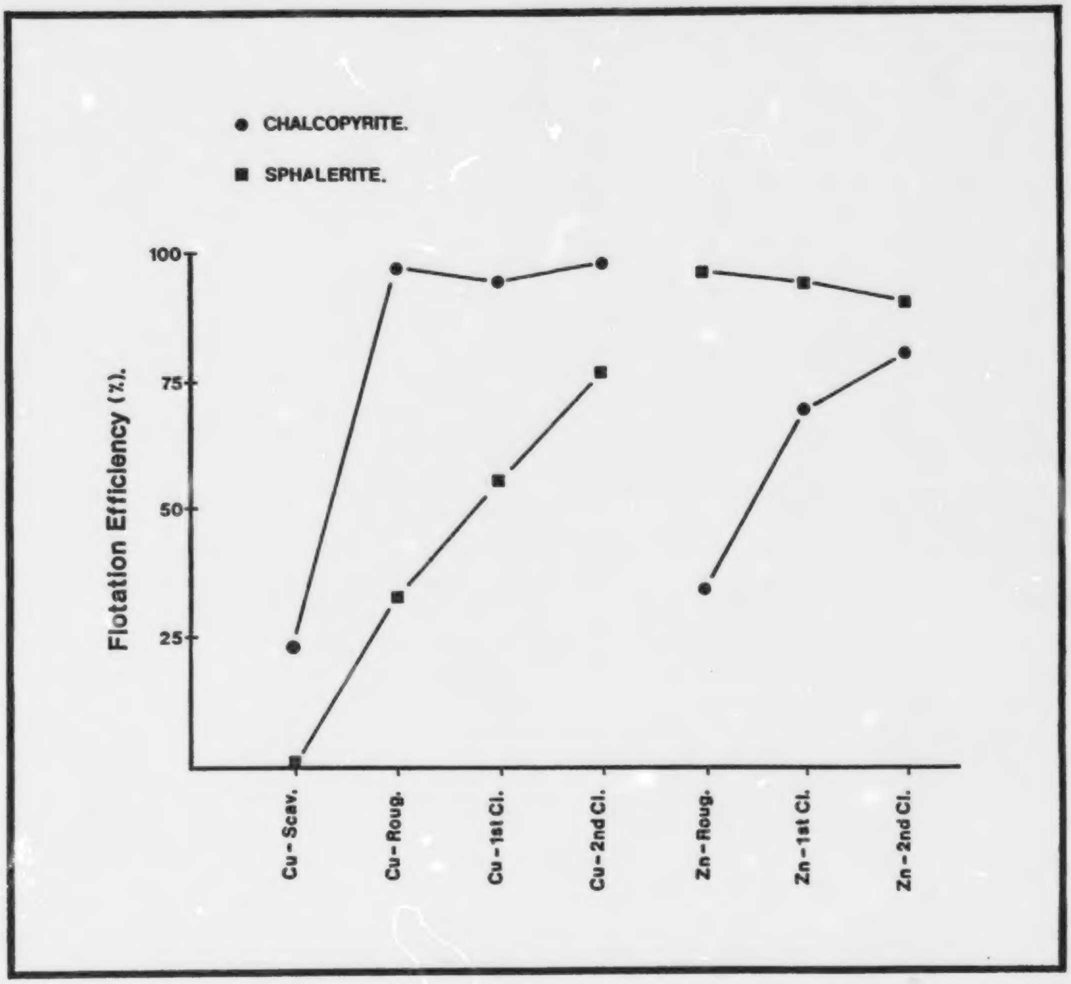


FIGURE 6.
PLOT OF FLOTATION EFFICIENCIES FOR FREE CHALCOPYRITE AND SPHALERITE.

CHALCOPYRITE

With the exception of the Cu-scavenger, all the units of the Cu-circuit exhibit chalcopyrite flotation efficiencies of greater than 95 %. The slightly lower efficiency of the Cu-1st cleaner reflects the stage where the most intense rejection of contaminants occurs, and where thus free chalcopyrite is also partly rejected. It is improbable that the Cu-scavenger, which by definition is 'pulling hard', would not recover a greater proportion of the free chalcopyrite presented to it, than the Cu-2nd cleaner. Because observations of apparent free grains inevitably occur Zn-circuit products, the chalcopyrite liberation in the Cu-scavenger tailings (the Zn-circuit feed) is adjusted to account for these apparent losses. Thus, this lower flotation efficiency for free chalcopyrite in the Cu-scavenger is at least partly spurious. The lower flotation efficiency may also reflect 'over-sized' free chalcopyrite rejected by the Cu-rougher, which form a large fraction of the free chalcopyrite in the Cu-scavenger feed, and which are rejected to the Zn-circuit. This however, is not supported by petrographic evidence, as 'over-sized' free chalcopyrite is not observed in the tailings and is relatively minor in the Zn-concentrate.

The flotation efficiencies of free chalcopyrite for the Zn-circuit units, are based on values of adjusted % free chalcopyrite that all fall below 12 %, and are therefore potentially subject to strong biases by observations of apparent free grains derived from the dominant assemblage of unliberated grains in the constituent products. The recovery of free grains of chalcopyrite in each concentrate of the Zn-circuit is less than 1 %, and confirms the insignificance of these potentially biased observations. Therefore, the behaviour of apparent free grains of chalcopyrite in the Zn-circuit is interpreted to be largely spurious.

SPHALERITE

The flotation efficiencies for free sphalerite through the Cu-circuit steadily increase to 77.3 % at the Cu-2nd cleaner. This confirms that a fraction of the free sphalerite in the feed is recovered through the Cu-circuit, with progressively lower rates of rejection. As in the Trout Lake circuit, the initial recovery of this free sphalerite does not occur at the Cu-scavenger, but at the Cu-rougher stage. Nonetheless, 77.3 % of the free sphalerite presented to the Cu-2nd cleaner is recovered in the concentrate. While this is higher than at the Trout Lake Cu-2nd cleaner (i.e. 70.5 %), it may merely reflect that the flotation units are 'pulling hard', rather than activation or significantly poorer selective flotation of the sphalerite relative to the Trout Lake circuit.

The flotation efficiencies for free sphalerite through the Zn-circuit drop from essentially 96.6 % to 90.7 % at the Zn-2nd cleaner. Thus, the recovery of free sphalerite grains entering the Zn-circuit is high with progressive rejection in the Zn-cleaning stages. The 96.6 % efficiency of the Zn-rougher translates into 2.88 % loss of free sphalerite to the tailings. Minor medium-grained, free sphalerite was observed petrographically in the tailings (See Plate 2H). Nonetheless, the activation of sphalerite by $CuSO_4$, and the flotation using the Minerec collector is effective in the recovery of free sphalerite within the Zn-circuit (i.e. 96.6 %).

MATERIAL BALANCES OF THE UNLIBERATED GRAINS

The raw data tables and plots related to this section of the study, comprise Parts C and D (i.e. pages 12 to 93) of Volume 2, the appendix of raw image analysis data. In this section, the modal fraction of chalcopyrite and sphalerite that occur as grains locked with other minerals (i.e. the % unliberated), and the classification of these unliberated grains, are evaluated. The classification of chalcopyrite and sphalerite occurring in particles containing at least one grain, of one or more of any of the other minerals (e.g. chalcopyrite Vs all) were determined for each product, and material balance calculations were done on these data.

The systematic stereological bias that yields observations of apparent free grains derived from actually unliberated grains of the mineral of interest, generates slightly elevated liberations of that mineral (See PETROGRAPHIC AND IMAGE ANALSIS). By corollary the observed % unliberated is correspondingly depressed. Hill et al. (1986) note that this bias is minimized in assemblages consisting of complex particles, and that the window between actual and apparent data is approximately 15 % or less for a nominal particle assemblage. However, while the % unliberated may be depressed, the compositional integrity or classification of the observed unliberated grains is essentially preserved, and theoretically requires no correction (See Hill et al. 1986 and Healy & Petruk 1989 for further discussion).

The raw % unliberated data for chalcopyrite and sphalerite in each of the 20 mill product samples is presented in Table 2 of appendix A, and are condensed for the 17 mill products in Table 21. As was done with the % free data, the modal fractions occurring in selected composite classes of the mineral of interest were first weighted in terms of the amount of the respective mineral present in each product. These data were transformed by MATBAL into a coherent set of adjusted modal fractions of the mineral, and recoveries of the same were calculated. The % unliberated chalcopyrite and sphalerite, extracted from these material balances, are presented in Table 22. The adjusted values for % free and % unliberated chalcopyrite, for example, are based on (1) the adjusted modal quantities of chalcopyrite, (2) the adjusted modal fraction of free chalcopyrite, and (3) the adjusted modal fraction of unliberated chalcopyrite; three independently material balanced variables. In Table 3 of appendix A the calculated % free and % unliberated were combined for each of the two minerals in each of the products, so as to assess the degree of convergence of the adjustments (i.e. ideally if the adjustments approach the true values, the sum should equal 100 %). Values of 93.0 % and 90.5 % combined free and unliberated chalcopyrite in the Cu- and Zn-concentrates, respectively, are anomalous, and reflect the deviation of the adjusted variables from the true values. From Table 3 of appendix A, it is apparent that the convergence is generally poor (i.e. > 5 % of 100) for chalcopyrite (e.g. 93.8 % in the feed), but is satisfactory for sphalerite (e.g. 98.8 % in the feed). Despite the poor material balance modelling of chalcopyrite liberations through the circuit, with approximately 6 % of the chalcopyrite in the feed being unaccounted for on recombining the adjusted composite classes, the adjusted data is considered adequate.

The recoveries of the % unliberated chalcopyrite and sphalerite, are presented in Table 23. While 69 % of the chalcopyrite in the feed is unliberated, 85.0 % of this is recovered in the Cu-concentrate, with 1.68 % reporting to the Zn-concentrate and 13.3 % to the tailings. Thus, while 97.7 % of the free grains are recovered, an additional 85.0 % of the unliberated grains are also recovered in the Cu-concentrate. The remaining unliberated grains report principally to the tailings (principally as pyrite-chalcopyrite and gangue-chalcopyrite particles; See Plate 2C), with minor recoveries to the Zn-concentrate (principally as granular chalcopyrite-sphalerite, chalcopyrite-pyrite or chalcopyrite-pyrrhotite particles, as diablastic chalcopyrite-sphalerite particles, and as chalcopyrite-disease; See Plate 2F, G and H).

While 70.9 % of the sphalerite in the feed is unliberated, 67.4 % of this sphalerite is recovered in the Zn-concentrate, with 12.7 % lost to the tailings. The remaining 19.9 % is recovered in the Cu-concentrate, and is slightly greater than the recovery of free sphalerite in the Cu-concentrate (i.e. 19.4 %). While the 19.4 % recovery of free sphalerite in the Cu-concentrate is anomalous, the slightly higher recovery of unliberated sphalerite is essentially indistinguishable from the former within the uncertainties of the analysis, though it may be partly attributable to composite particles of chalcopyrite and sphalerite. The recoveries/losses of 13.3 % and 12.7 % of unliberated chalcopyrite and sphalerite, respectively to the tailings, represents greater than 98 % and 95 % of the chalcopyrite and sphalerite, respectively in the tailings, indicating that the principal losses of chalcopyrite and sphalerite to the tailings are essentially liberation dependent.

TABLE 21

CONDENSED RAW % UNLIBERATED MINERAL IN THE 17 HBMS MILL PRODUCTS.

PRODUCT.	CHALCOPYRITE	SPHALERITE
1. Feed.	82.3	84.1
2. Cu-Scav Co.	65.6	36.2
3. Cu-Roug Co.	80.2	67.1
4. Cu-Cl.1 Fd.	79.5	69.0
5. Cu-Cl.1 Co.	66.8	56.4
6. Cu.Cl.2 Co.	43.6	39.8
7. Zn-Roug Fd.	95.0	79.8
8. Zn-Roug Co.	87.9	64.8
9. Zn-Cl.1 Co.	91.1	58.0
10. Zn-Cl.2 Co.	78.0	46.3
11. Cu-Roug Tl.	92.6	83.7
12. Cu-Scav Tl.	94.7	67.8
13. Cu-Cl.1 Tl.	92.6	54.1
14. Cu-Cl.2 Tl.	91.9	57.3
15. Zn-Cl.1 Tl.	93.2	75.2
16. Zn-Cl.2 Tl.	80.0	54.0
17. Zn-Roug Tl.	96.7	94.4

TABLE 22

ADJUSTED % UNLIBERATED MINERAL IN THE 17 HBMS MILL PRODUCTS.

PRODUCT.	CHALCOPYRITE	SPHALERITE
1. Feed.	69.1	70.9
2. Cu-Scav Co.	53.1	35.7
3. Cu-Roug Co.	70.9	68.8
4. Cu-Cl.1 Fd.	71.7	67.7
5. Cu-Cl.1 Co.	67.0	69.9
6. Cu.Cl.2 Co.	65.9	72.2
7. Zn-Roug Fd.	95.3	70.9
8. Zn-Roug Co.	83.9	67.8
9. Zn-Cl.1 Co.	80.0	66.4
10. Zn-Cl.2 Co.	79.7	67.3
11. Cu-Roug Tl.	92.4	70.5
12. Cu-Scav Tl.	95.5	70.6
13. Cu-Cl.1 Tl.	96.2	64.9
14. Cu-Cl.2 Tl.	91.9	58.1
15. Zn-Cl.1 Tl.	91.7	75.3
16. Zn-Cl.2 Tl.	81.7	53.3
17. Zn-Roug Tl.	97.9	95.3

NOTE : Calculated above values by dividing the material balance adjusted modal fraction of unliberated chalcopyrite by the adjusted modal abundances of chalcopyrite, both of which datasets were calculated independently by MATBAL.

TABLE 23

RECOVERY OF UNLIBERATED MINERALS IN THE 17 HBMS MILL PRODUCTS.

PRODUCT.	CHALCOPYRITE	SPHALERITE
1. Feed.	100.0	100.0
2. Cu-Scav Co.	0.45	0.14
3. Cu-Roug Co.	109.8	35.2
4. Cu-Cl.1 Fd.	115.2	38.3
5. Cu-Cl.1 Co.	90.4	23.0
6. Cu.Cl.2 Co.	85.0	19.9
7. Zn-Roug Fd.	16.0	85.7
8. Zn-Roug Co.	2.72	73.0
9. Zn-Cl.1 Co.	1.91	71.1
10. Zn-Cl.2 Co.	1.68	67.4
11. Cu-Roug Tl.	15.4	80.3
12. Cu-Scav Tl.	15.0	80.2
13. Cu-Cl.1 Tl.	24.8	15.4
14. Cu-Cl.2 Tl.	5.43	3.13
15. Zn-Cl.1 Tl.	1.04	5.54
16. Zn-Cl.2 Tl.	0.24	3.63
17. Zn-Roug Tl.	13.3	12.7

CHALCOPYRITE

The raw liberation and classification data for chalcopyrite in each of the 20 mill product samples (compiled from raw chalcopyrite Vs all classifications) are given in Table 4 of appendix A, are condensed for the 17 mill products in Table 24, and are reduced into principal composite classes in Table 25 (See also Figs. 1 to 4 in appendix B). Material balances were done on these data, rather than on those of the individual classes, so as to (1) avoid poor balances in individual classes with poor counting statistics, (2) examine only those classes with relatively uniform flotation characteristics, and (3) reduce the number of material balances and hence detail of the evaluation.

Before proceeding to deal with the results of the material balance calculations of unliberated chalcopyrite, it is worth examining the raw liberation and classification data for chalcopyrite in the feed (See Table 25). While 17.7 % of the chalcopyrite is free, another 48.4 % occurs in the 70-99.9 classes (i.e. sub-free), 9.18 % in the middling and 24.7 % sub-middling classes (See Fig. 7A). Thus, 66.1 % of the chalcopyrite occurs as free or sub-free grains and are nominally expected to be recovered in the Cu-concentrate. The 9.18 % middling grains of chalcopyrite are expected to be partly recovered in the Cu-concentrate, while the losses in this category are potentially amenable to regrinding. However, the 24.7 % sub-middling grains are expected to be largely rejected to the tailings, and are nominally not amenable to regrinding. Thus, from the raw data, approximately 30 % of the chalcopyrite in the feed is expected to be unrecoverable (to the Cu-concentrate) and rejected to the tailings under relatively standard flotation conditions. The proportions of these composite classes are altered on recombining the material balances of each composite class, though the overall pattern is essentially retained (See Table 29).

The percent of chalcopyrite in each composite class in Table 25 were multiplied by the adjusted modal quantities of chalcopyrite in the respective product to give the modal fraction of chalcopyrite in each composite class (See Table 26). The results of the material balance calculations on these data are (1) the adjusted modal fractions of chalcopyrite in the composite classes in each product (See Table 27), and (2) the simultaneously calculated recoveries of these composite classes in each product (See Table 28). The data in the columns entitled 'Adjusted Cu Assay', 'Adjusted Cpy Modal Amounts', 'Recovery of Cpy', '% Free' and '% Locked' have all been introduced in previous sections or chapters, and are presented here for ease of comparison and completeness.

It can be seen from Table 28 that 97.7 % of the free and 85.0 % of the unliberated chalcopyrite is recovered in the Cu-concentrate, resulting in a corresponding 89.2 % chalcopyrite recovery (See Fig. 7C). The recoveries of free and sub-free chalcopyrite (i.e. 97.7 % and 98.7 %, respectively) are indistinguishable within the uncertainties of the analysis. These recoveries, as well as that of middling grains (i.e. 92.2 %) are exceptionally high, and contribute significantly to the high overall recovery of chalcopyrite. The principal losses occur in the sub-middling classes with only 34.1 % recovered in the Cu-concentrate, 4.89 % reporting to the Zn-concentrate and 61.0 % to the tailings.

The recovery of chalcopyrite in terms of the adjusted modal fraction of chalcopyrite in free, unliberated or in composite classes of unliberated grains in each product is given in Table 29 and graphically presented in Figure 9. This data is not generated directly by material balance calculation, but is extracted by multiplying the recovery of chalcopyrite in a product by the adjusted modal fraction of chalcopyrite occurring in the specific class of free and/or unliberated chalcopyrite. Thus, the value of 70.7 % in the Cu-2nd cleaner concentrate under the '70-100' composite class implies that 70.7 % of the 89.2 % chalcopyrite recovery (i.e. 79.3 %) is attributable to free and sub-free or 'recoverable' classes of chalcopyrite.

However, significant cross-balancing discrepancies become apparent on recombining the independently material balance adjusted modal fractions of chalcopyrite in each composite class for a given product. By summing the values in the 'Free', '70-99.9', '30-70' and '0-30' composite classes in Table 29, totals of 90.0, 80.8, 1.30 and 8.61 % are derived for the feed, Cu-concentrate, Zn-concentrate and tailings, respectively. These values correspond to approximately 90 %, rather than 100 %, of the chalcopyrite

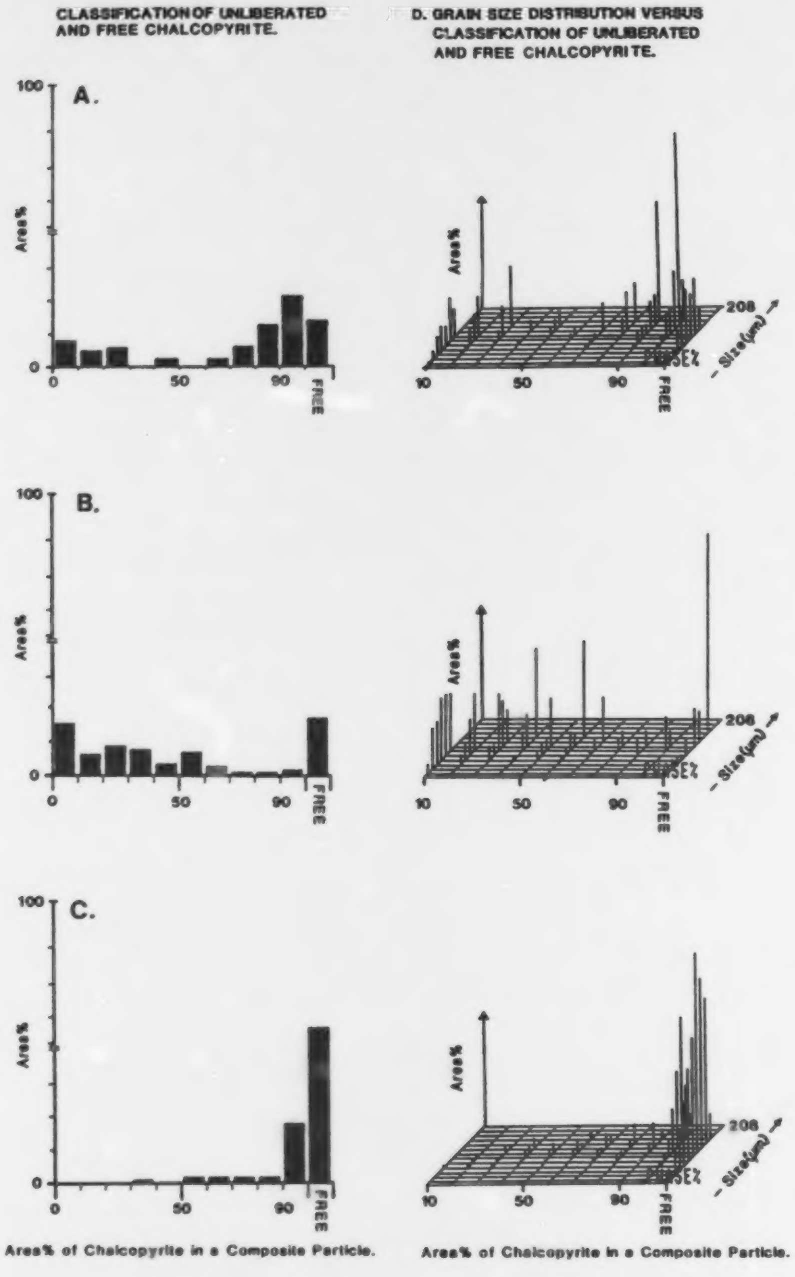


FIGURE 7.
PLOTS OF THE RAW CHALCOPYRITE CLASSIFICATION IN (A) THE FLOAT FEED, (B) THE Zn-CONCENTRATE, AND (C) THE Cu-CONCENTRATE.

TABLE 24

CONDENSED RAW CLASSIFICATION OF FREE AND UNLIBERATED CHALCOPYRITE IN THE 17 HBMS MILL PRODUCTS.

PRODUCT.	ADJUSTED Cu ASSAY.	% LOCKED	CLASSIFICATION OF UNLIBERATED GRAINS.										% FREE
		(0-99.9)	0-10	10-20	20-30	30-40	40-50	50-60	60-70	70-80	80-90	90-99	(99.9-100)
1. Feed	1.00	82.3	10.4	6.60	7.69	1.34	3.22	1.40	3.22	7.40	15.5	25.5	17.7
2. Cu-Scav Co.	5.60	61.4	5.53	6.34	6.30	3.03	2.37	3.01	3.01	2.83	3.82	25.2	38.6
3. Cu-Roug Co.	8.40	78.0	3.65	4.61	3.40	4.49	2.07	3.71	2.44	3.51	8.30	41.8	22.0
4. Cu-Cl.1 Fd.	8.00	79.5	3.59	5.78	4.73	4.43	3.18	3.06	4.52	4.21	11.7	34.2	20.5
5. Cu-Cl.1 Co.	14.9	66.8	1.49	2.10	2.32	2.28	2.44	3.56	2.77	3.22	6.69	39.9	33.2
6. Cu-Cl.2 Co.	17.4	43.6	0.71	1.39	1.64	2.61	1.78	3.17	3.02	3.19	3.65	22.5	56.4
7. Zn-Roug Fd.	0.30	94.9	58.3	19.8	12.0	0.30	----	1.53	2.87	----	----	0.12	5.05
8. Zn-Roug Co.	1.10	87.9	21.5	16.7	9.97	10.8	3.48	10.4	4.91	2.87	0.72	6.54	12.1
9. Zn-Cl.1 Co.	1.10	91.1	19.2	15.8	12.9	15.4	6.89	4.81	5.44	0.28	2.16	8.10	8.91
10. Zn-Cl.2 Co.	1.10	78.0	19.2	8.66	11.4	10.4	5.60	9.81	4.60	2.16	2.53	3.72	22.0
11. Cu-Roug Tl.	0.30	94.9	48.2	11.4	3.39	9.76	1.47	7.08	3.30	1.15	6.66	0.20	7.36
12. Cu-Scav Tl.	0.30	94.7	33.4	17.6	10.8	18.1	11.6	0.41	0.57	0.85	----	1.44	5.29
13. Cu-Cl.1 Tl.	2.10	92.6	16.1	21.9	15.4	12.1	9.19	3.96	3.29	3.75	4.02	2.83	7.37
14. Cu-Cl.2 Tl.	3.80	91.9	9.89	16.8	8.82	15.0	10.8	5.84	4.95	1.11	5.04	13.7	8.07
15. Zn-Cl.1 Tl.	1.20	93.2	27.7	29.7	5.42	6.21	0.57	0.35	5.08	4.56	7.07	6.52	6.82
16. Zn-Cl.2 Tl.	1.40	80.0	22.7	9.64	15.6	2.32	5.66	2.07	0.38	----	0.90	20.7	20.0
17. Zn-Roug Tl.	0.20	96.7	45.3	25.7	9.56	14.5	1.10	----	----	----	----	0.52	3.29

TABLE 25

CONDENSED RAW CLASSIFICATION OF FREE AND UNLIBERATED CHALCOPYRITE REDUCED INTO THE PRINCIPAL COMPOSITE CLASSES IN THE 17 HBMS MILL PRODUCTS.

PRODUCT.	ADJUSTED Cu ASSAY.	% FREE (99.9-100)	% LOCKED (0-99.9)	COMPOSITE CLASSES OF MIDDLING AND SUB-MIDDLING.			COMPOSITE CLASSES OF FREE AND SUB-FREE.	
				0-30	30-70	0-70	70-99.9	70-100
1. Feed.	1.10	17.7	82.3	24.7	9.18	33.9	48.4	66.1
2. Cu-Scav Co.	5.57	34.4	65.6	18.2	11.4	29.5	36.1	70.5
3. Cu-Roug Co.	8.36	19.8	80.2	11.7	12.7	24.9	55.8	75.6
4. Cu-Cl.1 Fd.	8.00	20.5	79.5	14.1	15.2	29.4	50.1	70.6
5. Cu-Cl.1 Co.	15.0	33.2	66.8	5.91	11.1	17.0	49.8	82.8
6. Cu-Cl.2 Co.	17.4	56.4	43.6	3.74	10.6	14.3	30.8	85.7
7. Zn-Roug Fd.	0.25	5.04	95.0	90.1	4.70	94.8	0.12	5.16
8. Zn-Roug Co.	1.12	12.1	87.9	48.2	29.6	77.8	10.1	22.2
9. Zn-Cl.1 Co.	1.15	8.91	91.1	47.9	32.5	80.5	10.5	19.5
10. Zn-Cl.2 Co.	1.10	22.0	78.0	39.3	30.4	69.7	8.41	30.4
11. Cu-Roug Tl.	0.26	7.36	92.6	63.0	21.6	84.6	8.01	15.4
12. Cu-Scav Tl.	0.25	5.29	94.7	61.8	30.7	92.5	2.29	7.58
13. Cu-Cl.1 Tl.	2.10	7.37	92.6	53.4	28.5	82.0	10.6	18.0
14. Cu-Cl.2 Tl.	3.78	8.07	91.9	35.5	36.6	72.1	19.9	27.9
15. Zn-Cl.1 Tl.	1.15	6.82	93.2	62.8	12.2	75.0	18.2	25.0
16. Zn-Cl.2 Tl.	1.36	20.0	80.0	47.9	10.4	58.4	21.6	41.6
17. Zn-Roug Tl.	0.23	3.29	96.7	80.6	15.6	96.2	0.52	3.81

TABLE 26

CONDENSED RAW MODAL FRACTION OF FREE AND UNLIBERATED CHALCOPYRITE REDUCED INTO THE PRINCIPAL COMPOSITE CLASSES IN THE 17 HBMS MILL PRODUCTS.

PRODUCT.	ADJUSTED Cpy MODAL AMOUNTS.	% FREE (99.9-100)	% LOCKED (0-99.9)	COMPOSITE CLASSES OF MIDDLING AND SUB-MIDDLING. 0-30	30-70	0-70	COMPOSITE CLASSES OF FREE AND SUB-FREE. 70-99.9	70-100
1. Feed.	2.41	0.48	2.23	0.67	0.25	0.92	1.31	1.79
2. Cu-Scav Co.	7.91	2.19	4.18	1.16	0.73	1.88	2.30	4.49
3. Cu-Roug Co.	22.7	4.26	17.3	2.52	2.73	5.36	12.0	16.3
4. Cu-Cl.1 Fd.	21.7	4.21	16.3	2.90	3.12	6.04	10.3	14.5
5. Cu-Cl.1 Co.	39.8	11.9	23.9	2.12	3.98	6.10	17.9	29.7
6. Cu-Cl.2 Co.	46.2	23.4	18.1	1.55	4.39	5.93	12.8	35.5
7. Zn-Roug Fd.	0.29	0.01	0.27	0.25	0.01	0.27	0.00	0.02
8. Zn-Roug Co.	2.19	0.26	1.90	1.04	0.64	1.68	0.21	0.48
9. Zn-Cl.1 Co.	2.12	0.18	1.88	0.99	0.67	1.66	0.22	0.40
10. Zn-Cl.2 Co.	2.23	0.48	1.70	0.86	0.66	1.52	0.18	0.66
11. Cu-Roug Tl.	0.29	0.03	0.34	0.23	0.08	0.31	0.03	0.06
12. Cu-Scav Tl.	0.27	0.01	0.25	0.16	0.08	0.24	0.01	0.02
13. Cu-Cl.1 Tl.	6.41	0.48	6.02	3.47	1.85	5.33	0.69	1.17
14. Cu-Cl.2 Tl.	10.0	0.81	9.24	3.57	3.68	7.25	1.99	2.80
15. Zn-Cl.1 Tl.	2.13	0.15	3.99	1.34	0.26	3.21	0.39	1.07
16. Zn-Cl.2 Tl.	1.55	0.31	1.25	0.75	0.16	0.91	0.34	0.65
17. Zn-Roug Tl.	0.24	0.01	0.21	0.18	0.03	0.21	0.00	0.01

TABLE 27

ADJUSTED MODAL FRACTION OF CHALCOPYRITE OCCURRING IN FREE, UNLIBERATED AND IN COMPOSITE CLASSES OF CHALCOPYRITE IN THE 17 HBMS MILL PRODUCTS.

PRODUCT.	ADJUSTED Cpy MODAL AMOUNTS.	% FREE (99.9-100)	% LOCKED (0-99.9)	COMPOSITE CLASSES OF MIDDLING AND SUB-MIDDLING. 0-30	30-70	0-70	COMPOSITE CLASSES OF FREE AND SUB-FREE. 70-99.9	70-100
1. Feed.	2.41	0.59	1.66	0.29	0.22	0.54	1.07	1.71
2. Cu-Scav Co.	7.91	2.30	4.20	1.16	0.74	1.88	2.32	4.87
3. Cu-Roug Co.	22.7	5.40	16.1	2.78	2.90	5.68	9.70	15.7
4. Cu-Cl.1 Fd.	21.7	5.03	15.6	2.84	2.96	5.80	9.09	14.7
5. Cu-Cl.1 Co.	39.8	10.5	26.7	2.37	4.28	6.46	19.1	30.7
6. Cu-Cl.2 Co.	46.2	12.5	30.4	2.14	4.42	6.31	22.7	36.6
7. Zn-Roug Fd.	0.29	0.02	0.28	0.21	0.02	0.28	0.02	0.02
8. Zn-Roug Co.	2.19	0.21	1.84	1.07	0.51	1.77	0.25	0.58
9. Zn-Cl.1 Co.	2.12	0.25	1.70	0.89	0.58	1.50	0.21	0.48
10. Zn-Cl.2 Co.	2.23	0.24	1.78	0.91	0.66	1.61	0.18	0.45
11. Cu-Roug Tl.	0.29	0.02	0.27	0.20	0.02	0.26	0.02	0.03
12. Cu-Scav Tl.	0.27	0.01	0.26	0.20	0.02	0.26	0.02	0.02
13. Cu-Cl.1 Tl.	6.41	0.45	6.16	3.23	1.84	5.24	0.70	1.17
14. Cu-Cl.2 Tl.	10.0	0.81	9.19	3.45	3.65	7.19	1.99	2.80
15. Zn-Cl.1 Tl.	2.13	0.16	1.95	1.33	0.24	2.04	0.36	0.81
16. Zn-Cl.2 Tl.	1.55	0.28	1.26	0.76	0.16	0.92	0.34	0.62
17. Zn-Roug Tl.	0.24	0.01	0.24	0.19	0.01	0.24	0.01	0.01

TABLE 28

RECOVERY OF CHALCOPYRITE OCCURRING IN FREE, UNLIBERATED AND IN COMPOSITE CLASSES OF CHALCOPYRITE IN THE 17 HBMS MILL PRODUCTS.

PRODUCT.	RECOVERY OF Cpy.	% FREE (99.9-100)	% LOCKED (0-99.9)	COMPOSITE CLASSES OF MIDDLING AND SUB-MIDDLING. 0-30	30-70	0-70	COMPOSITE CLASSES OF FREE AND SUB-FREE. 70-99.9	70-100
1. Feed.	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0
2. Cu-Scav Co.	0.57	0.69	0.45	0.71	0.59	0.62	0.39	0.51
3. Cu-Roug Co.	06.9	103.0	109.8	108.0	147.3	119.4	103.0	103.7
4. Cu-Cl.1 Fd.	111.0	104.3	115.2	119.7	163.5	132.5	104.9	105.3
5. Cu-Cl.1 Co.	93.2	99.1	90.4	45.8	108.3	67.5	100.5	100.7
6. Cu-Cl.2 Co.	89.2	97.7	85.0	34.1	92.2	54.4	98.7	99.1
7. Zn-Roug Fd.	11.6	2.50	16.0	69.9	8.77	49.0	1.63	1.30
8. Zn-Roug Co.	2.24	0.87	2.72	8.95	5.60	8.03	0.57	0.83
9. Zn-Cl.1 Co.	1.65	0.78	1.91	5.70	4.86	5.19	0.37	0.52
10. Zn-Cl.2 Co.	1.45	0.63	1.68	4.89	4.64	4.67	0.27	0.41
11. Cu-Roug Tl.	11.4	. 96	15.4	66.6	8.40	46.3	1.71	1.39
12. Cu-Scav Tl.	10.9	2.27	15.0	65.9	7.81	45.6	1.32	0.88
13. Cu-Cl.1 Tl.	17.8	5.23	24.8	73.9	55.1	65.0	4.36	4.57
14. Cu-Cl.2 Tl.	4.07	1.34	5.43	11.6	16.1	13.1	1.83	1.60
15. Zn-Cl.1 Tl.	0.79	0.23	1.04	4.06	0.96	3.36	0.30	0.42
16. Zn-Cl.2 Tl.	0.20	0.15	0.24	0.80	0.23	0.53	0.10	0.11
17. Zn-Roug Tl.	9.40	1.64	13.3	61.0	3.18	41.0	1.06	0.47

recovered in each of these products, and indicates a systematic error in the balances. Thus, approximately 10 % of the chalcopyrite in the feed is not accounted for on modelling the behaviour of the constituent composite classes of chalcopyrite through the circuit. This lack of convergence of the material balance adjustments (to the raw experimental values) with the 'true' values, is unsatisfactory. Interpretation is consequently subject to the uncertainties inherent in these adjusted data. The errors appear to be concentrated in the material balance of the sub-middling composite class, where the only substantial adjustment to the raw data occurred (See Table 25 and 29).

From the first row of data in Table 29, corresponding to the float feed, 12.0 % of the chalcopyrite is attributed to sub-middling grains and 9.13 % to middling grains. An additional 44.4 % of the chalcopyrite is attributed to sub-free grains, 24.5 % to free grains, while the remaining 10 % is unaccounted for on recombining the material balances. The data in Table 29 for the four principal composite classes (i.e. free, sub-free, middling and sub-middling) were normalized on the basis of the sum of these classes equalling 100 % in the feed (i.e. normalization factor = 1.107). This is justified to the extent that the raw data are implicitly normalized on the feed, which if adjusted significantly, introduces significant cross-balancing discrepancies. It is noteworthy that the feed is typically a poor and unrepresentative sample, and generally incurs major adjustments because of the poor counting statistics for the mineral of interest i.e. low modal abundances (W. Petruk, pers. comm. 1988). These normalized data are given in Table 30, and are the primary data upon which the behaviour of chalcopyrite through the circuit is based.

From the first row of data in Table 30, corresponding to the feed, 27.2 % occurs as free, 49.3 % as sub-free, 10.1 % as middling and 13.3 % as sub-middling grains of chalcopyrite (See Fig. 9). Thus, 76.5 % of the chalcopyrite in the feed occurs in 'recoverable' classes, whereas 24.4 % occurs in middling and sub-middling classes, and are potentially subject to high losses to the tailings, but are in part amenable to regrinding. Of the 76.5 % of the chalcopyrite in the feed that occurs in 'recoverable' classes 75.5 % are recovered (i.e. 98.7 % efficient) in the Cu-concentrate. The minor losses to the Zn-concentrate and tailings are insignificant within the resolution of the analysis, and limits of acceptable losses. The coarse-grained (i.e. 'over-sized') free and sub-free chalcopyrite lost to the Zn-concentrate (i.e. less than 0.3 % of the chalcopyrite in the feed) is potentially recoverable through regrinding of an appropriate preceeding stream. Coarse-grained chalcopyrite, whether free or partly unliberated, is likely to concentrate in the Cu-1st cleaner tailings (as indicated by petrographic examination), and for additional reasons given below is recommended for regrinding. Nonetheless, the overall recovery of free and sub-free chalcopyrite is excellent, and leaves negligible latitude for measurable improvement.

Losses to the Zn-concentrate and tailings of 37 % of the combined middling and sub-middling chalcopyrite correspond to 9.5 % of the chalcopyrite in the feed. The losses of middling and sub-middling grains (to the tailings) account for 91 % of the chalcopyrite in the tailings (See Fig. 8A). The large proportion of chalcopyrite in the feed occurring as sub-middling grains (i.e. 13.3 %) confirms that poor liberation is achieved, and that it is the principal source of the substantial chalcopyrite losses to the tailings. Sub-middling grains are not readily amenable to regrinding, as the individual fine-grained chalcopyrite grain forms less than 30 % of the particle. It thus requires intensive grinding of the particle to liberate the fine chalcopyrite. However, many of these grains occur as intertsitial films on, and wedges between, clusters of pyrite grains (See Plates 1A, B and 2G), and are expected to be readily liberated during regrinding. Regrinding of the middlings seems unjustified in terms of Cu recovery (but not concentrate grade) as 92.2 % are recovered in the Cu-concentrate. Only 4.64 % and 3.18 % of the middlings are lost to the Zn-concentrate and tailings, respectively, constituting 33 % and 5 %, respectively of the chalcopyrite recovered in the respective products. Thus, the minor amount of middling grains in the feed (i.e. 10.1 % of the chalcopyrite), and the high recovery in the Cu-concentrate (i.e. 92.2 %) due to the 'hard pulling' circuit, militate against significant improvement in Cu recovery by regrinding these grains. Nonetheless, approximately 1 % of the chalcopyrite is lost to the tailings as middling grains, and are potentially amenable to recovery through regrinding. The latter would also tend to increase the concentrate grade.

The principal losses of chalcopyrite to the Zn-concentrate and tailings are as middling and particularly sub-middling grains (See Fig. 9). The sub-middling grains constitute 46 % and 88 %, respectively of the losses to these products (See Figs. 7C and 8A). Significant improvement in Cu recovery would necessitate increased liberation from these grains through regrinding. Unlike in the Trout Lake circuit (See Healy & Petruk 1989), the Cu-scavenger concentrate is depleted in middling and sub-middling grains, and is not recommended for regrinding. However, as in the Trout Lake circuit, the Cu-1st cleaner tailings is recommended for regrinding; being preferentially enriched in these classes of chalcopyrite. This minor stream (i.e. flow rate of 6.68) recovers 73.9 % of the sub-middling and 55.1 % of the middling chalcopyrite, accounting for 32 % and 56 %, respectively of the chalcopyrite in the stream. The product contains 46.3 % gangue, 42.2 % pyrite, 6.41 % chalcopyrite and 3.77 % sphalerite, and is thus enriched in pyrite, chalcopyrite and sphalerite relative to the Cu-rougher tailings (stream flow rate of 96.2); a more conventional choice for regrinding (Compare Figs. 8B and C). Furthermore, the contained gangue and pyrite is largely locked with chalcopyrite, whereas that in the Cu-rougher tailings is largely free, and in which the recovery of middling and sub-middling chalcopyrite is 8.40 % and 66.6 %, respectively. The Cu-1st cleaner tailings offers a much more selective target for regrinding in that the material amenable to and/or selected for regrinding is preferentially concentrated in a smaller stream. However, as only 8 % of the middling chalcopyrite, the principal class amenable to regrinding, is lost to the Zn-concentrate and tailings, the Cu recovery is not likely to be significantly increased without recourse to fine-regrinding to address the sub-middling chalcopyrite. Thus, the selection of the Cu-1st cleaner tailings permits a much lower grind to be achieved, and hence the significant fraction of sub-middling grains to be addressed, and a higher overall liberation to be achieved in the ground product.

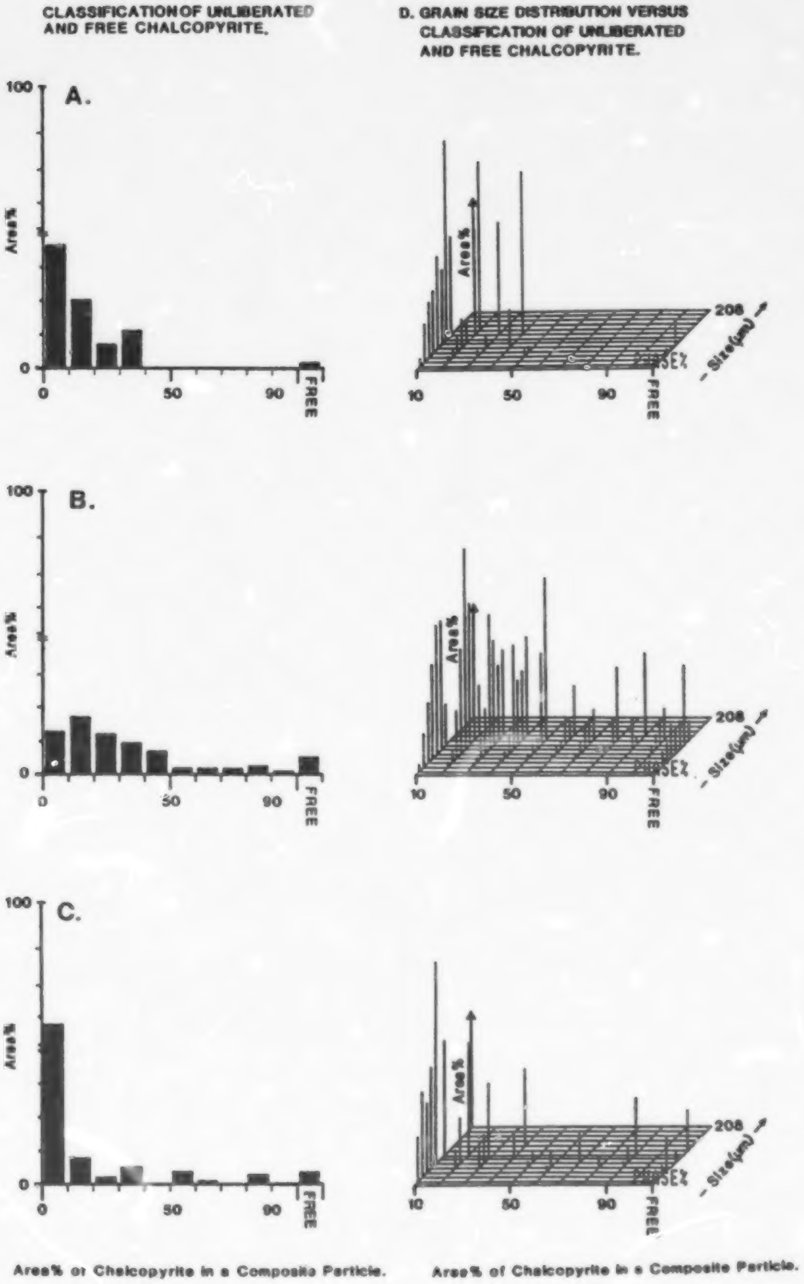


FIGURE 8.
PLOTS OF THE RAW CHALCOPYRITE CLASSIFICATION IN (A) THE TAILINGS, (B) THE Cu-1st CLEANER TAILINGS, AND (C) THE Cu-ROUGHER TAILINGS.

TABLE 29

THE MODAL FRACTION OF THE CHALCOPYRITE RECOVERY OCCURRING IN COMPOSITE CLASSES OF CHALCOPYRITE IN THE 17 HBMS MILL PRODUCTS.

PRODUCT.	RECOVERY OF Cpy.	% FREE (99.9-100)	% LOCKED (0-99.9)	COMPOSITE CLASSES OF MIDDLING AND SUB-MIDDLING. 0-30	30-70	0-70	COMPOSITE CLASSES OF FREE AND SUB-FREE. 70-99.9	70-100
1. Feed.	100.0	24.5	68.9	12.0	9.13	22.4	44.4	71.0
2. Cu-Scav Co.	9.57	0.17	0.30	0.08	0.05	0.14	0.17	0.35
3. Cu-Roug Co.	106.9	25.4	75.8	13.1	13.7	26.8	45.7	73.9
4. Cu-Cl.1 Fd.	111.0	25.7	79.8	14.5	15.1	29.7	46.5	75.2
5. Cu-Cl.1 Co.	93.2	24.6	62.5	5.55	10.0	15.1	44.7	71.9
6. Cu-Cl.2 Co.	89.2	24.1	58.7	4.13	8.85	12.2	43.8	70.7
7. Zn-Roug Fd.	11.6	0.08	11.2	8.40	0.80	11.2	0.80	0.80
8. Zn-Roug Co.	2.24	0.22	1.88	1.09	0.52	1.81	0.26	0.59
9. Zn-Cl.1 Co.	1.65	0.20	1.32	0.69	0.45	1.17	0.16	0.37
10. Zn-Cl.2 Co.	1.45	0.16	1.16	0.59	0.43	1.05	0.12	0.29
11. Cu-Roug Tl.	11.4	0.79	10.6	7.86	0.79	10.2	0.79	1.18
12. Cu-Scav Tl.	10.9	0.40	10.5	8.07	0.81	10.5	0.81	0.81
13. Cu-Cl.1 Tl.	17.8	1.25	17.1	8.97	5.11	14.6	1.94	3.25
14. Cu-Cl.2 Tl.	4.07	0.33	3.74	1.40	1.49	2.93	0.81	1.14
15. Zn-Cl.1 Tl.	0.79	0.06	0.72	0.49	0.09	0.76	0.13	0.30
16. Zn-Cl.2 Tl.	0.20	0.04	0.16	0.10	0.02	0.12	0.04	0.08
17. Zn-Roug Tl.	9.40	0.39	9.40	7.44	0.39	9.40	0.39	0.39

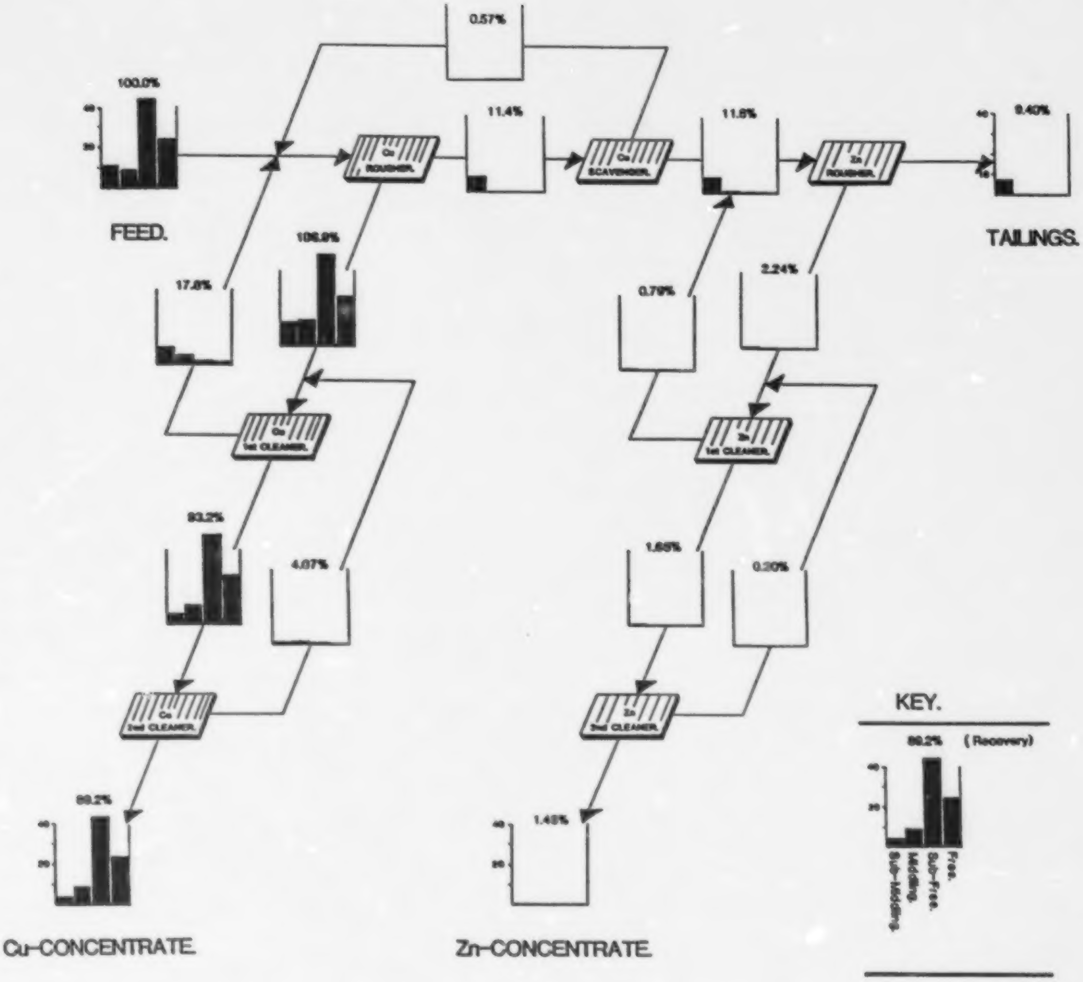

0.57%
100.0%
40
20
FEED.
Cu ROUGHER.
11.4%
Cu SCAVENGER.
11.8%
Zn ROUGHER.
9.40%
40
10
TAILINGS.
17.8%
106.9%
0.79%
2.24%
Cu 1st CLEANER.
Zn 1st CLEANER.
93.2%
4.07%
1.65%
0.20%
Cu 2nd CLEANER.
Zn 2nd CLEANER.
89.2%
40
20
Cu-CONCENTRATE.
1.48%
40
20
Zn-CONCENTRATE.
KEY.
89.2% (Recovery)
40
20
Sub-Middling.
Middling.
Sub-Free.
Free.

FIGURE 9.
THE MODAL FRACTION OF THE CHALCOPYRITE RECOVERY MODELLED IN TERMS OF THAT OCCURRING AS FREE, SUB-FREE, MIDDLING AND SUB-MIDDLING GRAINS IN THE 17 HBMS MILL PRODUCTS.

TABLE 30

THE NORMALIZED MODAL FRACTION OF THE CHALCOPYRITE RECOVERY OCCURRING IN COMPOSITE CLASSES OF CHALCOPYRITE IN THE 17 HBMS MILL PRODUCTS.

PRODUCT.	Free	Sub-Free	Middling	Sub-Middling	Cpy Recovery
1. Feed.	27.2	49.3	10.1	13.3	100.0
2. Cu-Scav Co.	0.19	0.19	0.06	0.09	0.5
3. Cu-Roug Co.	28.2	50.8	15.2	14.6	108.8
4. Cu-Cl.1 Fd.	28.5	51.7	16.8	16.1	113.1
5. Cu-Cl.1 Co.	27.3	49.7	11.1	6.17	94.2
6. Cu.Cl.2 Co.	26.8	48.7	9.83	4.59	89.9
7. Zn-Roug Fd.	0.09	0.89	0.89	9.33	11.2
8. Zn-Roug Co.	0.24	0.29	0.58	1.21	2.3
9. Zn-Cl.1 Co.	0.22	0.18	0.50	0.77	1.7
10. Zn-Cl.2 Co.	0.18	0.13	0.48	0.66	1.5
11. Cu-Roug Tl.	0.88	0.88	0.88	8.73	11.4
12. Cu-Scav Tl.	0.44	0.90	0.90	8.96	11.2
13. Cu-Cl.1 Tl.	1.39	2.16	5.68	9.96	19.2
14. Cu-Cl.2 Tl.	0.37	0.90	1.66	1.56	4.5
15. Zn-Cl.1 Tl.	0.07	0.14	0.10	0.54	0.9
16. Zn-Cl.2 Tl.	0.04	0.04	0.02	0.11	0.2
17. Zn-Roug Tl.	0.43	0.43	0.43	8.26	9.6

Note: 'Cpy Recovery' is the sum of the values for the four classes in that product, and is ideally equivalent to the calculated recovery of Chalcopyrite (total) in that product.

SPHALERITE

The raw liberation and classification data for sphalerite in each of the 20 mill product samples (compiled from raw sphalerite Vs all classifications) are given in Table 5 of appendix A, are condensed for the 17 mill products in Table 31 (See also Figs. 5 to 8 in appendix B), and are reduced into principal composite classes in Table 32. As with chalcopyrite, material balances were only done on these data, rather than on those of the individual classes.

Before proceeding to deal with the results of the material balance calculations of unliberated sphalerite, it is worth examining the raw liberation and classification data for sphalerite in the feed (See Table 32). While 15.9 % of the sphalerite is free, another 47.0 % occurs in the sub-free, 12.6 % in the middling and 24.5 % in the sub-middling classes (See Fig. 10A). Thus, 62.9 % of the sphalerite occurs as free or sub-free grains and are nominally expected to be recovered in the Zn-concentrate. The 12.6 % middling grains of sphalerite are expected to be partly recovered in the Zn-concentrate, while the losses in this category are potentially amenable to regrinding. However, the 24.5 % sub-middling grains are expected to be largely rejected to the tailings, and nominally are not amenable to regrinding. Thus, from the raw data, approximately 30 % of the sphalerite in the feed is expected to be unrecoverable (to the Zn-concentrate) and possibly rejected to the tailings under relatively standard flotation conditions. The proportions of these composite classes are however altered on recombining the material balances of each composite class (See Table 36).

The percent of sphalerite in each composite class in Table 32 were multiplied by the adjusted modal abundances of sphalerite (total) in the respective product to give the modal fraction of sphalerite in each composite class (See Table 33). The results of the material balance calculations on these data are (1) the adjusted modal fractions of sphalerite in the composite classes in each product (See Table 34), and (2) the simultaneously calculated recoveries of these composite classes in each product (See Table 35). The data in the columns entitled 'Adjusted Zn Assay', 'Adjusted Sphal Modal Amounts', 'Recovery of Sphal', '% Free' and '% Locked' have all been introduced in previous sections or chapters, and are presented here for ease of comparison and completeness.

It can be seen from Table 35 that 77.7 % of the free and 67.4 % of the unliberated sphalerite are recovered in the Zn-concentrate (See Fig. 10C), resulting in a corresponding 71.0 % sphalerite and 40.2 % Zn recovery. The discrepancy in the recoveries of free and sub-free sphalerite (i.e. 77.7 % and 83.1 %, respectively) is probably not significant within the uncertainties of the analysis. These recoveries are anomalously poor, and contribute significantly to the low overall recovery of sphalerite. The principal losses in each of the four composite classes are of: free grains (i.e. 19.4 %), sub-free grains (i.e. 14.6 %) and middling grains (i.e. 29.9 %) to the Cu-concentrate; and of sub-middling grains (i.e. 53.1 %) to the tailings. The three former classes comprise the bulk of the 19.5 % losses of sphalerite to the Cu-concentrate (See Fig. 10B), whereas the sub-middling classes comprise the bulk of the 9.5 % losses of sphalerite to the tailings (See Fig. 11A). Whereas the losses of sphalerite to the Cu-concentrate are relatively uniform amongst the four composite classes, those to the tailings increase sharply from an approximate level of 2.5 % for the free and sub-free classes to 14.6 % for middling and 53.1 % for the sub-middling. Thus, the losses to the Cu-concentrate are apparently largely liberation independent, whereas those to the tailings are largely liberation dependent.

Because of significant preceeding losses of sphalerite in the Cu-circuit, only 80.5 % of the sphalerite in the feed enters the Zn-circuit. The losses to the Cu-concentrate of 19.4 % of the free, 14.6 % of the sub-free, 29.9 % of the middling and 20.1 % of the sub-middling sphalerite, dictate that the Cu-scavenger tailings, the feed to the Zn-circuit, contains 80.6 % of the free, 85.5 % of the sub-free, 70.1 % of the middling and 79.9 % of the sub-middling sphalerite (See Table 35). Thus, the recoveries in the Zn-concentrate of 77.7 %, 83.1 %, 55.4 % and 26.7 % of these classes, respectively, correspond

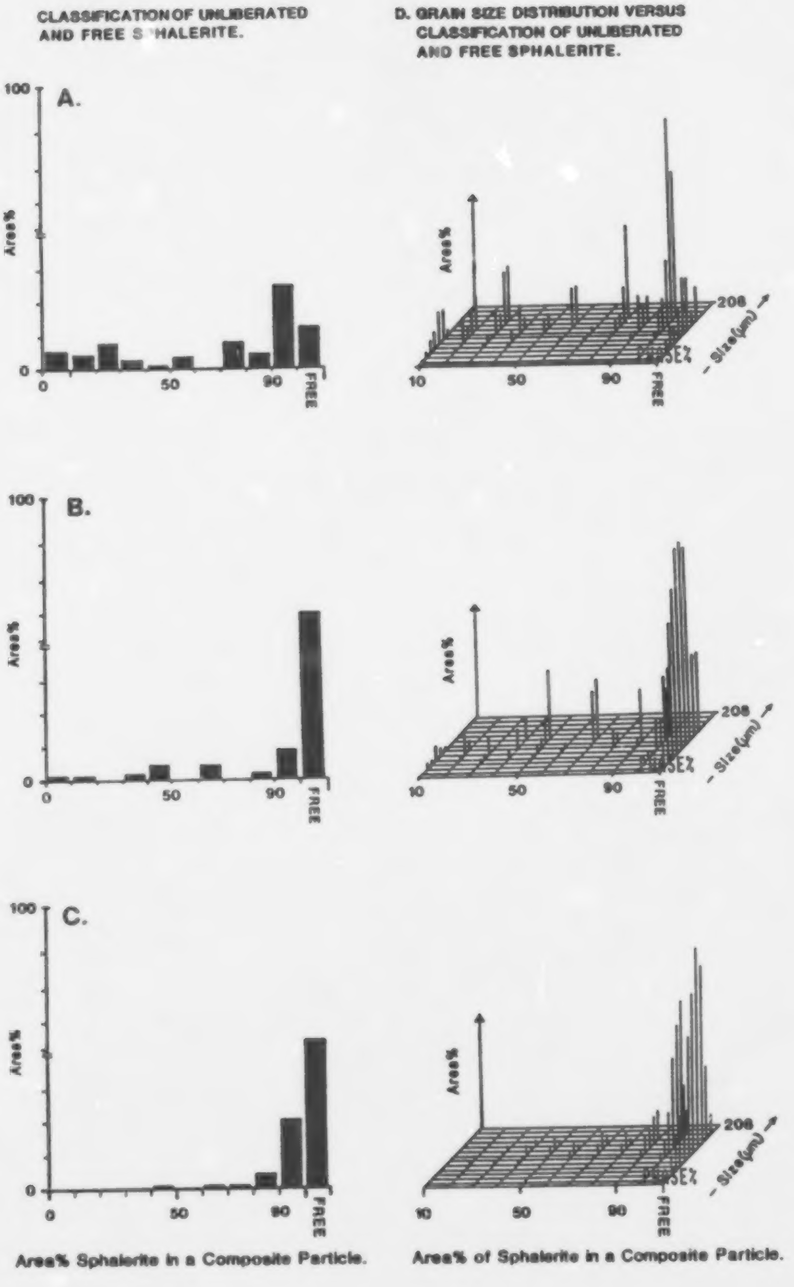


FIGURE 10.
PLOTS OF THE RAW SPHALERITE CLASSIFICATION IN (A) THE FLOAT FEED, (B) THE Cu-CONCENTRATE AND (C) THE Zn-CONCENTRATE.

TABLE 31

CONDENSED RAW CLASSIFICATION OF FREE AND UNLIBERATED SPHALERITE IN THE 17 HBMS MILL PRODUCTS.

PRODUCT.	ADJUSTED Zn ASSAY.	% LOCKED	CLASSIFICATION OF UNLIBERATED GRAINS.										% FREE
		(0-99.9)	0-10	10-20	20-30	30-40	40-50	50-60	60-70	70-80	80-90	90-99	(99.9-100)
1. Feed	1.80	84.1	7.00	6.79	10.7	4.04	2.06	5.22	1.29	10.3	6.39	30.3	15.9
2. Cu-Scav Co.	5.10	37.1	4.38	2.56	3.22	1.20	1.50	1.86	0.51	2.44	8.68	10.4	62.9
3. Cu-Roug Co.	5.70	67.1	4.99	3.38	3.16	4.26	4.00	4.48	7.62	5.88	6.43	19.2	32.9
4. Cu-Cl.1 Fd.	5.80	69.0	5.41	3.37	0.19	5.16	3.71	1.62	6.65	2.22	3.93	36.8	31.0
5. Cu-Cl.1 Co.	7.40	56.4	4.75	2.18	2.34	7.99	5.53	6.61	2.40	2.46	4.77	17.4	43.6
6. Cu-Cl.2 Co.	7.50	39.8	2.40	2.34	1.79	3.07	6.23	1.01	6.15	1.91	3.81	11.1	60.2
7. Zn-Roug Fd.	1.60	68.3	51.0	0.29	----	----	----	----	15.8	----	1.26	----	31.7
8. Zn-Roug Co.	32.7	64.8	0.80	1.03	1.77	2.89	2.00	2.23	2.49	3.84	7.27	40.4	35.2
9. Zn-Cl.1 Co.	42.4	58.0	0.47	1.52	1.99	1.95	1.89	3.00	3.60	4.19	7.13	32.3	42.0
10. Zn-Cl.2 Co.	46.2	46.3	0.32	0.97	1.29	1.88	2.24	1.97	2.81	2.72	6.50	25.6	53.7
11. Cu-Roug Tl.	1.50	83.7	11.5	11.2	1.75	8.79	1.01	12.2	4.67	6.08	6.72	19.8	16.3
12. Cu-Scav Tl.	1.50	67.8	5.83	4.61	3.32	0.20	0.70	20.9	0.86	4.27	0.78	26.3	32.2
13. Cu-Cl.1 Tl.	4.50	54.1	3.20	2.46	3.20	4.11	2.60	6.57	2.73	8.23	5.43	15.6	45.9
14. Cu-Cl.2 Tl.	6.70	57.3	4.22	3.33	2.63	2.52	8.47	3.62	2.36	0.60	0.90	28.6	42.7
15. Zn-Cl.1 Tl.	1.20	75.2	5.51	6.29	10.3	11.8	6.73	11.6	4.20	5.17	5.69	7.85	24.8
16. Zn-Cl.2 Tl.	22.9	54.0	2.20	3.62	4.89	4.50	5.20	6.86	3.12	5.42	6.17	12.1	46..0
17. Zn-Roug Tl.	0.80	94.4	20.9	17.6	4.78	11.2	----	3.56	9.13	14.6	6.04	6.59	5.64

TABLE 32

CONDENSED RAW CLASSIFICATION OF FREE AND UNLIBERATED SPHALERITE REDUCED INTO THE PRINCIPAL COMPOSITE CLASSES IN THE 17 HBMS MILL PRODUCTS.

PRODUCT.	ADJUSTED Zn ASSAY.	% FREE (99.9-100)	% LOCKED (0-99.9)	COMPOSITE CLASSES OF MIDDLING AND SUB-MIDDLING.			COMPOSITE CLASSES OF FREE AND SUB-FREE.	
				0-30	30-70	0-70	70-99.9	70-100
1. Feed.	1.81	15.9	84.1	24.5	12.6	37.1	47.0	62.9
2. Cu-Scav Co.	5.05	63.8	36.2	10.1	5.47	15.7	21.5	84.4
3. Cu-Roug Co.	5.71	32.9	67.1	11.6	20.3	31.9	35.2	68.1
4. Cu-Cl.1 Fd.	5.79	31.0	69.0	8.97	17.1	26.1	43.0	74.0
5. Cu-Cl.1 Co.	7.36	43.6	56.4	9.27	22.5	31.8	24.6	68.2
6. Cu-Cl.2 Co.	7.51	60.2	39.8	6.53	16.5	28.0	16.8	77.0
7. Zn-Roug Fd.	1.59	20.2	79.8	20.2	16.6	36.8	43.0	63.2
8. Zn-Roug Co.	32.8	35.2	64.8	3.60	9.61	13.2	51.5	86.7
9. Zn-Cl.1 Co.	42.5	42.0	58.0	3.98	10.4	14.4	43.6	85.6
10. Zn-Cl.2 Co.	46.3	53.7	46.3	2.58	8.90	11.5	34.8	88.5
11. Cu-Roug Tl.	1.54	16.3	83.7	24.5	26.7	51.1	32.6	48.9
12. Cu-Scav Tl.	1.53	32.2	67.8	13.8	22.7	36.4	31.4	63.6
13. Cu-Cl.1 Tl.	4.47	45.9	54.1	8.86	16.0	24.9	29.3	75.2
14. Cu-Cl.2 Tl.	6.65	42.7	57.3	10.2	17.0	27.2	30.1	72.8
15. Zn-Cl.1 Tl.	8.69	24.8	75.2	22.1	34.3	56.4	18.7	43.5
16. Zn-Cl.2 Tl.	23.0	46.0	54.0	10.7	19.7	30.4	22.7	68.7
17. Zn-Roug Tl.	0.78	5.64	94.4	43.3	23.9	67.2	27.2	32.9

TABLE 33

CONDENSED RAW MODAL FRACTION OF FREE AND UNLIBERATED SPHALERITE REDUCED INTO THE PRINCIPAL COMPOSITE CLASSES IN THE 17 HBMS MILL PRODUCTS.

PRODUCT.	ADJUSTED Sphal MODAL AMOUNTS.	% FREE (99.9-100)	% LOCKED (0-99.9)	COMPOSITE CLASSES OF MIDDLING AND SUB-MIDDLING.			COMPOSITE CLASSES OF FREE AND SUB-FREE.	
				0-30	30-70	0-70	70-99.9	70-100
1. Feed.	1.50	0.27	1.45	0.42	0.22	0.64	0.81	1.08
2. Cu-Scav Co.	2.39	1.50	0.85	0.34	0.13	0.37	0.51	1.98
3. Cu-Roug Co.	4.81	1.58	3.23	0.56	0.98	1.53	1.69	3.28
4. Cu-Cl.1 Fd.	4.89	1.52	3.38	0.44	0.84	1.28	2.11	3.63
5. Cu-Cl.1 Co.	6.22	2.66	3.44	0.57	1.37	1.94	1.50	4.16
6. Cu-Cl.2 Co.	6.30	3.71	2.45	0.40	1.02	1.42	1.04	4.74
7. Zn-Roug Fd.	1.34	0.31	1.23	0.31	0.26	0.57	0.66	0.97
8. Zn-Roug Co.	46.7	13.1	24.2	1.34	3.59	4.92	19.2	32.3
9. Zn-Cl.1 Co.	60.9	21.3	29.4	2.01	5.26	7.29	22.1	43.3
10. Zn-Cl.2 Co.	68.2	30.6	26.4	1.47	5.06	6.54	19.8	50.4
11. Cu-Roug Tl.	1.27	0.24	1.22	0.36	0.39	0.75	0.48	0.71
12. Cu-Scav Tl.	1.27	0.46	0.98	0.20	0.33	0.52	0.45	0.92
13. Cu-Cl.1 Tl.	3.77	1.74	2.05	0.34	0.61	0.94	1.11	2.85
14. Cu-Cl.2 Tl.	5.83	2.50	3.35	0.60	0.99	1.59	1.76	4.26
15. Zn-Cl.1 Tl.	8.88	2.24	6.78	1.99	3.09	5.08	1.69	3.92
16. Zn-Cl.2 Tl.	23.8	11.1	13.0	2.58	4.75	7.33	5.47	16.6
17. Zn-Roug Tl.	0.15	0.02	0.35	0.16	0.09	0.25	0.10	0.12

to recoveries of 96.4 %, 97.2 %, 79.0 % and 33.4 %, respectively of the grains in the corresponding composite class that enter the Zn-circuit. The recovery of the free and sub-free (i.e. recoverable) sphalerite that enters the Zn-circuit is thus high, and indicates that Zn flotation is operating relatively efficiently.

The recovery of sphalerite in terms of the adjusted modal fraction of sphalerite in free, unliberated or in composite classes of unliberated grains in each product is given in Table 36, and graphically presented in Figure 12. This data is not generated directly by material balance calculation, but is extracted by multiplying the recovery of sphalerite in a product by the adjusted modal fraction of sphalerite occurring in the specific class of free and/or unliberated sphalerite. Thus, the value of 55.9 % in the Zn-2nd cleaner concentrate under the '70-100' composite class implies that 55.9 % out of the 71.0 % sphalerite recovery (i.e. 78.7 %) is attributable to free and sub-free or 'recoverable' classes of sphalerite.

Significant cross-balancing discrepancies become apparent on recombining the independently material balanced adjusted modal fractions of sphalerite in each composite class for a given product. By summing the values in the 'Free', '70-99.9', '30-70' and '0-30' composite classes in Table 36, totals of 98.0, 18.7, 67.2 and 12.0 % are derived for the feed, Cu-concentrate, Zn-concentrate and tailings, respectively. These values correspond to 98, 96, 95 and 125 %, respectively, rather than 100 %, of the sphalerite recovered in each of these products, indicating significant errors in the individual balances. Thus, 2 % of the sphalerite in the feed is not accounted for on modelling the behaviour of the constituent composite classes of sphalerite through the circuit. In addition, the material balance model of the individual composite classes calculated losses to the tailings which exceed those when calculated as total sphalerite, with a corresponding deficit in the recoveries to the Zn-concentrate. This lack of convergence of the material balance adjusted values with the 'true' values, is however largely acceptable. The data interpretations are nonetheless subject to the uncertainties inherent in these adjusted data.

From the first row of data in Table 36, corresponding to the float feed, 14.0 % of the sphalerite is attributed to sub-middling grains and 16.0 % to middling grains. An additional 40.0 % of the sphalerite is attributed to sub-free grains, 28.0 % to free grains, while the remaining 2 % is unaccounted for on recombining the material balances. Thus, the proportions of these classes in the feed are comparable to those of chalcopyrite, except for the higher fraction of middling sphalerite, particularly. As with chalcopyrite, the data in Table 36 for the four principal composite classes (i.e. free, sub-free, middling and sub-middling) were normalized on the basis of the sum of these classes equalling 100 % in the feed (i.e. normalization factor of 1.02). These normalized data are given in Table 37, and are the primary data upon which the behaviour of the composite classes of sphalerite through the circuit are evaluated.

From the first row of data in Table 37, corresponding to the feed, 28.6 % occurs as free, 40.8 % as sub-free, 16.3 % as middling and 14.3 % as sub-middling grains of sphalerite (See Fig. 12). Thus, 69.4 % of the sphalerite in the feed occurs in free and sub-free (i.e. 'recoverable') classes, whereas

TABLE 34

ADJUSTED MODAL FRACTION OF SPHALERITE OCCURRING IN FREE, UNLIBERATED AND IN COMPOSITE CLASSES OF SPHALERITE IN THE 17 HBMS MILL PRODUCTS.

PRODUCT.	ADJUSTED Sphal MODAL AMOUNTS.	% FREE (99.9-100)	% LOCKED (0-99.9)	COMPOSITE CLASSES OF MIDDLING AND SUB-MIDDLING. 0-30	30-70	0-70	COMPOSITE CLASSES OF FREE AND SUB-FREE. 70-99.9	70-100
1. Feed.	1.50	0.42	1.07	0.21	0.24	0.45	0.60	1.04
2. Cu-Scav Co.	2.39	1.59	0.85	0.23	0.12	0.35	0.49	2.10
3. Cu-Roug Co.	4.81	1.46	3.31	0.58	1.02	1.61	1.64	3.23
4. Cu-Cl.1 Fd.	4.89	1.53	3.31	0.58	1.02	1.61	1.65	3.31
5. Cu-Cl.1 Co.	6.22	1.86	4.34	0.84	1.42	2.30	1.86	3.38
6. Cu-Cl.2 Co.	6.30	1.75	4.55	0.89	1.51	2.45	1.88	3.76
7. Zn-Roug Fd.	1.34	0.37	0.95	0.19	0.20	0.40	0.55	0.94
8. Zn-Roug Co.	46.7	14.1	31.8	2.95	6.39	9.87	21.0	35.8
9. Zn-Cl.1 Co.	60.9	19.2	40.5	3.35	7.73	11.8	27.6	47.7
10. Zn-Cl.2 Co.	68.2	20.8	45.9	3.51	8.32	12.7	31.9	53.7
11. Cu-Roug Tl.	1.27	0.36	0.90	0.17	0.17	0.36	0.54	0.91
12. Cu-Scav Tl.	1.27	0.35	0.90	0.17	0.17	0.36	0.54	0.91
13. Cu-Cl.1 Tl.	3.77	1.26	2.45	0.36	0.68	1.03	1.47	2.86
14. Cu-Cl.2 Tl.	5.83	2.40	3.39	0.59	1.00	1.58	1.77	4.19
15. Zn-Cl.1 Tl.	8.88	2.22	6.69	1.95	2.95	4.84	1.70	3.91
16. Zn-Cl.2 Tl.	23.8	11.2	12.7	2.54	4.70	7.17	5.45	16.6
17. Zn-Roug Tl.	0.15	0.01	0.15	0.12	0.04	0.15	0.02	0.03

TABLE 35

RECOVERY OF SPHALERITE OCCURRING IN FREE, UNLIBERATED AND IN COMPOSITE CLASSES OF SPHALERITE IN THE 17 HBMS MILL PRODUCTS.

PRODUCT.	RECOVERY OF Sphal.	% FREE (99.9-100)	% LOCKED (0-99.9)	COMPOSITE CLASSES OF MIDDLING AND SUB-MIDDLING. 0-30	30-70	0-70	COMPOSITE CLASSES OF FREE AND SUB-FREE. 70-99.9	70-100
1. Feed.	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0
2. Cu-Scav Co.	0.28	0.69	0.15	0.20	0.09	0.14	0.15	0.36
3. Cu-Roug Co.	36.3	39.5	35.2	32.0	49.2	40.4	31.0	35.1
4. Cu-Cl.1 Fd.	40.1	45.2	38.3	34.8	53.3	43.9	33.9	39.1
5. Cu-Cl.1 Co.	23.1	25.1	23.0	23.0	34.1	28.6	17.5	20.7
6. Cu-Cl.2 Co.	19.5	19.4	19.9	20.2	29.9	25.1	14.6	16.8
7. Zn-Roug Fd.	85.7	85.3	85.7	88.3	81.2	84.4	88.0	86.6
8. Zn-Roug Co.	76.3	82.4	73.0	35.2	66.6	53.4	85.6	84.0
9. Zn-Cl.1 Co.	75.9	85.9	71.1	30.5	61.5	48.7	85.9	85.6
10. Zn-Cl.2 Co.	71.0	77.7	67.4	26.7	55.4	43.9	83.1	80.7
11. Cu-Roug Tl.	80.8	81.3	80.3	80.1	70.2	75.0	85.6	83.6
12. Cu-Scav Tl.	80.5	80.6	80.2	79.9	70.1	74.9	85.5	83.2
13. Cu-Cl.1 Tl.	16.8	20.1	15.4	11.8	19.2	15.2	16.4	18.3
14. Cu-Cl.2 Tl.	3.82	5.64	3.13	2.84	4.17	3.44	2.90	3.96
15. Zn-Cl.1 Tl.	5.22	4.71	5.57	8.43	11.1	9.48	2.51	3.33
16. Zn-Cl.2 Tl.	4.83	8.18	3.63	3.77	6.10	4.83	2.77	4.86
17. Zn-Roug Tl.	9.50	2.88	12.7	53.1	14.6	31.0	2.35	2.52

30.6 % occurs in middling and sub-middling classes. The latter two composite classes are potentially subject to high losses to the tailings, but are in part amenable to regrinding. Thus, the 'effective' liberation of sphalerite (i.e. 69.4 %) is significantly less than for sphalerite in the Trout Lake feed (i.e. 91.5 %; See Healy & Petruk 1989), and is expected to result in significant losses of sphalerite.

Of the sphalerite in the feed that occurs in 'recoverable' classes (i.e. 69.4 %), only 56 % is recovered in the Zn-concentrate, with 11.5 % reporting to the Cu-concentrate and 1.94 % to the tailings. These minor losses to the tailings are relatively insignificant within the resolution of the analysis, and the limits of routine and acceptable losses. However, the loss of 11.5 % 'recoverable' sphalerite to the Cu-concentrate is significant, as it represents 17 % of the 'recoverable' sphalerite, and particularly as it comprises approximately 60 % of the sphalerite in the Cu-concentrate. These losses are apparently independent of the state of liberation of the constituent grains, and therefore are not amenable to regrinding as a corrective measure. Thus, of the 29 % total losses of sphalerite to the Cu-concentrate and tailings, 11.5 % (i.e. 40 %) are as 'recoverable' grains recovered in the Cu-concentrate. These are attributed to either (1) activation of sphalerite, or (2) poor selective flotation of sphalerite by the Minerec collector. It is noteworthy that 86 % of the sphalerite in the Trout Lake Cu-concentrate consisted of 'recoverable' grains, and accounted for a sphalerite grade of 6.7 % sphalerite (Healy & Petruk 1989). In contrast, only 60 % of the sphalerite in the HBMS Cu-concentrate consists of 'recoverable' grains, and accounts for a sphalerite grade of 3.76 % sphalerite (See Table 34). The higher recoveries of sphalerite in the low grade (with respect to chalcopyrite and sphalerite) HBMS Cu-concentrate, and the lower fraction of this sphalerite that occurs as 'recoverable' grains, compared to those of the Trout Lake Cu-concentrate, are consistent with poor selective flotation exaggerated by a 'hard pulling' circuit. Thus, the additional possible mechanism (by which sphalerite is recovered in the Cu-concentrate) of activation by abundant covellite and chalcocite in the HBMS circuit, which are relatively rare in the Trout Lake circuit, is a non-essential, though nonetheless possible mechanism.

Recoverable grains comprise 57.6 % of the sphalerite in the Cu-scavenger tailings, the feed to the Zn-circuit, of which 56 % (i.e. 97 %) are recovered in the Zn-concentrate. The remaining 12.6 % of the 69.2 % (accounted for) sphalerite recovered in the Zn-concentrate occurs as middling (i.e. 8.84 %) and sub-middling (i.e. 3.73 %) grains. Thus, of the 71.0 % sphalerite recovery in the Zn-concentrate, 79 % consists of 'recoverable' sphalerite. This contrasts with the 19.5 % sphalerite recovery in the Cu-concentrate, of which 60 % consists of 'recoverable' sphalerite. Thus, there is a proportionally larger fraction of sub-middling and particularly middling sphalerite in the Cu- rather than Zn-concentrate (See Figs. 10 and 12). This pattern is expected as the latter two classes are partly locked with chalcopyrite, and thus tend to report to the Cu-concentrate in amounts above those generated by the mechanism that recovers free and sub-free grains.

From Table 37, it is apparent that 1.94 % of the sphalerite in the feed is recovered in the tailings as 'recoverable' sphalerite. This constitutes a loss of 2.7 % of these specific classes of sphalerite, and a loss of 3.3 % of these that enter the Zn-circuit (i.e. 80.5 %). Thus, the Zn-circuit is 97 %

efficient in the recovery of 'recoverable' sphalerite presented to it. Because these classes comprise 69.4 % of the sphalerite in the feed, the loss of 3.3 % of these classes, constitutes 20 % (or less) of the sphalerite in the tailings. Because of the poor cross-balancing in the tailings (i.e. total = 12.3 % Vs actual adjusted value for total sphalerite of 9.5 %; See Table 37), the free and sub-free composite classes probably comprise approximately 15 % of the sphalerite in the tailings. Thus, approximately 85 % of the sphalerite in the tailings consists of middling (i.e. approx. 22 %) and sub-middling (i.e. approx. 63 %) grains. Despite the poor cross-balancing in the proportions of these classes in the tailings (due largely to 'rounding-off' errors on low values in the tailings), it is apparent that the bulk of the sphalerite lost to the tailings is unliberated, and occur in particles which do not behave as free sphalerite grains.

The recoveries of 55.4 % of the middling and 26.7 % of the sub-middling sphalerite in the Zn-concentrate (See Table 35) correspond to 8.84 and 3.73 %, respectively of the sphalerite in the feed (See Table 37), and together approximately 21 % of the sphalerite in the Zn-concentrate. Of the remaining middling sphalerite 29.9 % is recovered in the Cu-concentrate, corresponding to 4.67 % of the sphalerite in the feed. This high recovery of middling sphalerite to the Cu-concentrate probably reflects the high recovery of middling chalcopyrite to the Cu-concentrate (i.e. 92.2 %), with the consequent recovery of the bulk of the relatively common chalcopyrite-sphalerite middling particles. An additional 14.6 % of the middling sphalerite is lost to the tailings, and corresponds to 2.58 % of the sphalerite in the feed. Thus, total losses of 44.5 % of the middling sphalerite to the Cu-concentrate (i.e. 29.9 %) and tailings (i.e. 14.6 %), corresponding to 7.4 % of the sphalerite in the feed are potentially redressable by regrinding. This composite class, which is the most amenable to regrinding, exhibits the highest recoveries of the composite classes in the Cu-concentrate, and the highest circulating loads in the Cu-cleaner circuit, is preferentially enriched in the Cu-1st cleaner tailings.

The losses of 73.3 % of the sub-middling sphalerite to the Cu-concentrate (i.e. 20.2 %) and tailings (i.e. 53.1 %) correspond to 10.6 % of the sphalerite in the feed. These losses are only potentially redressable by extremely fine regrinding. However, as approximately 65 % of the sphalerite lost to the tailings occurs as sub-middling grains, of which only 11.8 % report to the Cu-1st cleaner tailings, only minor reduction in sphalerite losses to the tailings are expected from regrinding this stream. In contrast, only 14.6 % of the middling grains are lost to the tailings, and as 19.2 % of the middling grains report to the Cu-1st cleaner tailings, losses to the tailings due to middling sphalerite are expected to be substantially reduced through regrinding. Thus, the Cu-1st cleaner tailings, which exhibits the highest ratio of recovery (of middling and sub-middling classes) to stream flow rate, only contains 19.2 % and 11.8 % of these classes, respectively, corresponding to only 4.7 % of the sphalerite in the feed. In contrast, the Cu-rougher tailings, a stream commonly chosen for regrinding, contains 70.2 % of the middling and 80.1 % of the sub-middling sphalerite in the feed (Compare Figs. 11B and C), but because of the abundance of gangue and pyrite in this large stream (i.e. flow rate = 95.5), is unsuitable and resistant to the requisite fine grinding.

The principal fractions of sphalerite lost to the Cu-concentrate and tailings, and whose recovery to the Zn-concentrate would variably contribute to improved Zn recovery are (1) the 5.53 % loss as free and 5.94 % as sub-free grains reporting to the Cu-concentrate; (2) the 0.64 % loss as free and 1.30 % as sub-free grains reporting to the tailings (if the efficiency of sphalerite flotation in the Zn-circuit can be expected to exceed 97 %); (3) the 4.67 % loss as middling and 2.82 % as sub-middling grains reporting to the Cu-concentrate; and (4) the 2.58 % loss as middling and 7.76 % as sub-middling grains reporting to the tailings.

The largest fraction (i.e. free and sub-free grains reporting to the Cu-concentrate) corresponds to 11.5 % of the sphalerite in the feed, and is not amenable to regrinding; being esssentially liberation independent. The relatively high recovery of sub-middling chalcopyrite to the Cu-concentrate (i.e. 34.1 %), much of which is locked with sub-free grains of sphalerite, results from the 'hard pulling' setting of the circuit. This may possibly explain the high recovery of sub-free sphalerite to the Cu-concentrate. The second largest fraction (i.e. middling and sub-middling sphalerite reporting to the tailings), accounts for approximately 10 % of the sphalerite in the feed, consists principally of sub-middling sphalerite, and is thus only partly amenable to regrinding; if extremely fine. The third largest fractions (i.e. middling and sub-middling sphalerite reporting to the Cu-concentrate) corresponds to approximately 8 % of the sphalerite in the feed, and consists principally of middling sphalerite. Thus, approximately 18 % of the sphalerite in the feed that is lost to the Cu-concentrate and tailings is partly redressable by fine-regrinding. Thus, the maximum potential improvement in sphalerite recovery stemming from the application of intensive selective regrinding is 18 %, corresponding to approximately 9 % improvement in Zn-recovery.

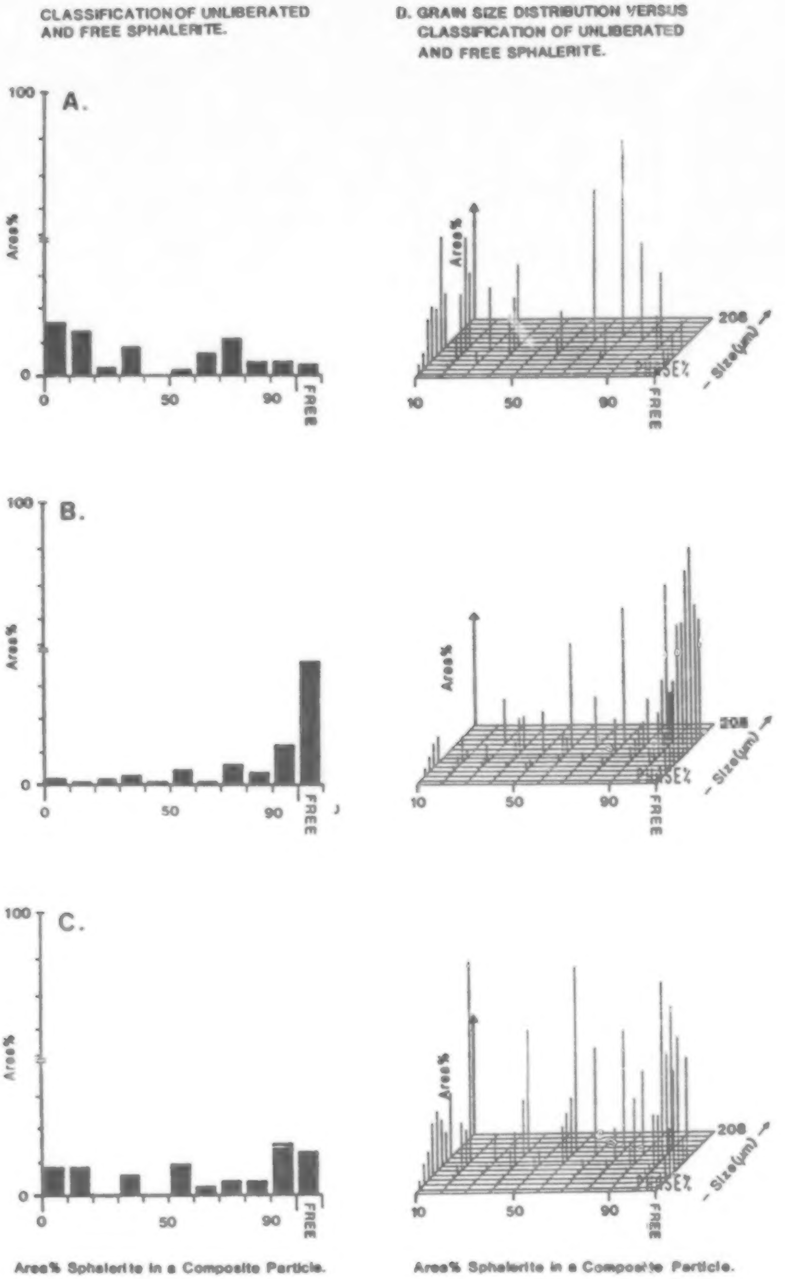


FIGURE 11.
PLOTS OF THE RAW SPHALERITE CLASSIFICATION IN (A) THE TAILINGS, (B) THE Cu-1st CLEANER TAILINGS, AND (C) THE Cu-ROUGHER TAILINGS.

TABLE 36

THE MODAL FRACTION OF THE SPHALERITE RECOVERY OCCURRING IN COMPOSITE CLASSES OF SPHALERITE IN THE 17 HBMS MILL PRODUCTS.

PRODUCT.	RECOVERY OF Sphal.	% FREE (99.9-100)	% LOCKED (0-99.9)	COMPOSITE CLASSES OF MIDDLING AND SUB-MIDDLING.			COMPOSITE CLASSES OF FREE AND SUB-FREE.	
				0-30	30-70	0-70	70-99.9	70-100
1. Feed.	100.0	28.0	71.3	14.0	16.0	30.0	40.0	69.3
2. Cu-Scav Co.	0.28	0.19	0.10	0.03	0.01	0.04	0.06	0.25
3. Cu-Roug Co.	36.3	11.0	25.0	4.38	7.70	12.2	12.4	24.4
4. Cu-Cl.1 Fd.	40.1	12.6	27.1	4.76	8.36	13.2	13.5	27.1
5. Cu-Cl.1 Co.	23.1	6.91	16.1	3.12	5.27	8.54	6.91	14.2
6. Cu-Cl.2 Co.	19.5	5.42	14.1	2.76	4.67	7.58	5.82	11.6
7. Zn-Roug Fd.	85.7	23.7	60.8	12.2	12.8	25.6	35.2	60.1
8. Zn-Roug Co.	76.3	23.0	52.0	4.82	10.4	16.1	34.3	58.5
9. Zn-Cl.1 Co.	75.9	23.9	50.5	4.18	9.63	14.7	34.4	59.5
10. Zn-Cl.2 Co.	71.0	21.7	47.8	3.65	8.66	13.2	33.2	55.9
11. Cu-Roug Tl.	80.8	22.9	57.3	10.8	10.8	22.9	34.4	57.9
12. Cu-Scav Tl.	80.5	22.2	57.1	10.8	10.8	22.8	34.2	57.7
13. Cu-Cl.1 Tl.	16.8	5.62	10.9	1.60	3.03	4.59	6.55	12.8
14. Cu-Cl.2 Tl.	3.82	1.57	2.22	0.39	0.66	1.04	1.16	2.75
15. Zn-Cl.1 Tl.	5.22	1.31	3.93	1.15	1.73	2.85	1.00	2.30
16. Zn-Cl.2 Tl.	4.83	2.27	2.58	0.52	0.95	1.46	1.11	3.37
17. Zn-Roug Tl.	9.50	0.63	9.50	7.60	2.53	9.50	1.27	1.90

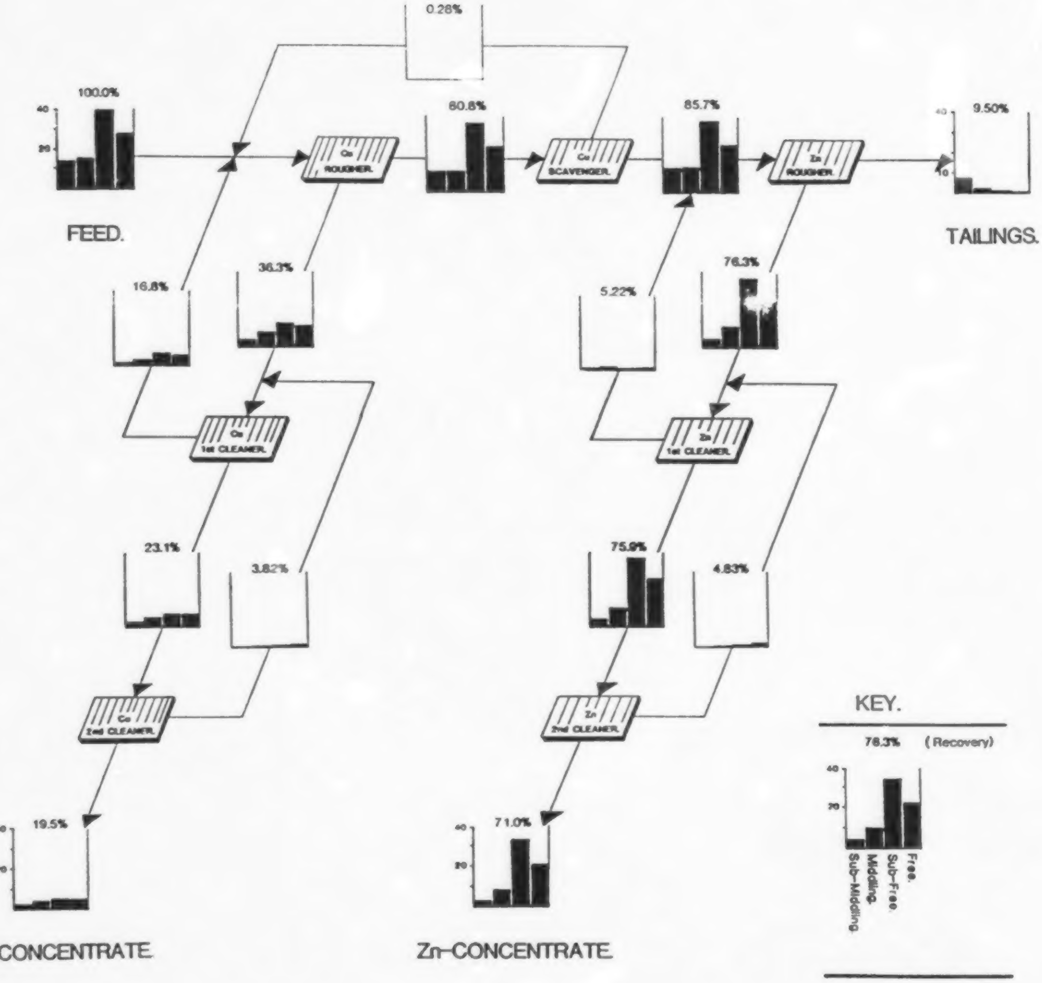


FIGURE 12.
THE MODAL FRACTION OF THE SPHALERITE RECOVERY MODELLED IN TERMS OF THAT OCCURRING AS FREE, SUB-FREE, MIDDLING AND SUB-MIDDLING GRAINS IN THE 17 HBMS MILL PRODUCTS.

TABLE 37

THE NORMALIZED MODAL FRACTION OF THE SPHALERITE RECOVERY OCCURRING IN COMPOSITE CLASSES OF SPHALERITE IN THE 17 HBMS MILL PRODUCTS.

PRODUCT.	Free	Sub-Free	Middling	Sub-Middling	Sphal Recovery
1. Feed.	28.6	40.8	16.3	14.3	100.0
2. Cu-Scav Co.	0.19	0.06	0.01	0.03	0.3
3. Cu-Roug Co.	11.2	12.7	7.86	4.47	36.2
4. Cu-Cl.1 Fd.	12.9	13.8	8.53	4.86	40.1
5. Cu-Cl.1 Co.	7.05	7.05	5.38	3.18	22.7
6. Cu.Cl.2 Co.	5.53	5.94	4.77	2.82	19.1
7. Zn-Roug Fd.	24.2	35.9	13.1	12.5	85.7
8. Zn-Roug Co.	23.5	35.0	10.6	4.92	74.0
9. Zn-Cl.1 Co.	24.4	35.1	9.83	4.27	73.6
10. Zn-Cl.2 Co.	22.1	33.9	8.84	3.73	68.6
11. Cu-Roug Tl.	23.4	35.1	11.0	11.0	80.5
12. Cu-Scav Tl.	22.7	34.9	11.0	11.0	79.6
13. Cu-Cl.1 Tl.	5.74	6.68	3.09	1.63	17.1
14. Cu-Cl.2 Tl.	1.60	1.18	0.67	0.40	3.9
15. Zn-Cl.1 Tl.	1.34	1.02	1.77	1.17	5.3
16. Zn-Cl.2 Tl.	2.32	1.13	0.97	0.53	5.0
17. Zn-Roug Tl.	0.64	1.30	2.58	7.76	12.3

Note: 'Sphal Recovery' is the sum of the values for the four classes in that product, and is ideally equivalent to the calculated recovery of sphalerite (total) in that product.

REGRIND

The strategy of a regrind is to employ a relatively coarse primary grind (with respect to the optimum grind of the valuable minerals), subsequently concentrate free and unliberated grains of the valuable minerals, and reject coarse gangue and pyrite prior to needless comminution. Subsequently, selected streams comprising a small proportion of the feed material and containing minor free gangue and pyrite, and with large middling (and sub-middling) fractions, may be reground to a size much finer than the primary grind, and thereby procure optimum liberation of the valuable minerals. This obviates (1) 'wasteful' grinding of free grains of valueless minerals, and (2) over-grinding free (or 'recoverable') grains of the valuable minerals. An additional benefit of regrinding is to 'freshen up' grains surfaces for further conditioning and flotation.

Streams commonly chosen for regrinding in the flotation circuits at Flin Flon are (1) Cu-rougher tailings; (2) Cu-rougher concentrate; and (3) Cu-1st cleaner tailings (B. Barlin, pers. comm. 1988). Healy & Petruk (1989) recommended regrinding the combined Cu-1st cleaner tailings and the Cu-scavenger concentrate in the Trout Lake circuit. This configuration is being implemented on the Trout Lake circuit at the time of writing. Thus, essentially four streams are considered when selecting a regrind in the HBMS circuit, but only with respect to chalcopyrite liberation. It becomes apparent below that no one choice of regrind stream is universally (i.e. with respect to chalcopyrite and sphalerite) applicable, such that two streams (e.g. the Cu- and Zn-1st cleaner tailings) should be reground (1) in separate mills or (2) combined in a single regrind mill. No regrind was being employed in the HBMS circuit at the time of sampling.

The principal fraction of chalcopyrite lost to the tailings and Zn-concentrate occurs as sub-middling and middling grains, is thus liberation dependent, and accounts for 9.8 % of the 10.8 % (i.e. 91 %) lost chalcopyrite. The principal fraction of lost sphalerite (i.e. 11.5 %) reports to the Cu-concentrate as 'recoverable' grains, is not liberation dependent, and hence is not amenable to regrinding. However, the two largest fractions of liberation dependent lost sphalerite (i.e. 10.3 % in the tailings and 7.5 % in the Cu-concentrate), occurring as middling and sub-middling grains, are amenable to regrinding. Thus, maximums of approximately 10 % of the chalcopyrite and 18 % of the sphalerite are potentially amenable to regrinding and recovery above present levels, and this would correspond to approximately 8 % and 9 % improvement in Cu and Zn recovery, respectively. However, because of the high proportion of these losses that occur as sub-middling grains, and because of the absence of a single suitable stream (for sphalerite particularly), it is improbable that such improvement in the recoveries of chalcopyrite and particularly sphalerite would be achieved, even on extremely fine regrinding to perhaps 80 % - 26 um.

Several of the relevant parameters for selecting a stream suitable for regrinding are tabulated for the five streams considered in Table 38. While the Cu-rougher tailings contains relatively high recoveries of middling and sub-middling sphalerite (accounting for 22 % of the sphalerite in the feed) it contains low modal quantities of chalcopyrite, low recoveries of middling chalcopyrite, and consists of 97 % gangue plus pyrite in a stream with a flow rate of 95.5. Regrinding this stream is only preferable to employing a finer primary grind in that (1) overgrinding of 'recoverable' chalcopyrite is avoided, and (2) grain surfaces are 'freshened up' for Cu-scavenging and Zn-conditioning. It has the distinct disadvantage of failing to address the middling chalcopyrite which would have pulled significant amounts of middling sphalerite (and pyrite) into the Cu-concentrate prior to that point in the circuit. It is therefore rejected as a regrind stream.

The Cu-scavenger concentrate is readily rejected on the basis of the low recoveries of middling and sub-middling chalcopyrite and sphalerite, accounting for 0.15 % and 0.04% of the chalcopyrite and sphalerite, respectively in the feed.

Comparison of the values in Table 38 for the Cu-rougher concentrate and the Cu-1st cleaner tailings show that the Cu-rougher concentrate excels in almost all of the parameter, particularly with respect to the fraction of the chalcopyrite and sphalerite recoveries occurring (critically) as middling and sub-middling grains (i.e. 29.8 Vs 15.6 % for chalcopyrite and 12.3 Vs 4.72 % for sphalerite). However, the comparison of stream flow rates is unfavourable (i.e. 11.3 Vs 6.68), though both streams are nonetheless small, and hence preferable to the Cu-rougher tailings. The principal disadvantage of regrinding the Cu-rougher concentrate is the potential for over-grinding chalcopyrite, with resulting losses to the Zn-circuit. It is for these latter two reasons only that the Cu-1st cleaner tailings is preferred to the Cu-rougher concentrate (or Cu-1st cleaner feed, which includes the Cu-2nd cleaner tailings). Intensifying rejection from the Cu-rougher concentrate by the Cu-1st cleaner would direct a greater mass of middling and sub-middling fractions of chalcopyrite and sphalerite, and 'over-sized' chalcopyrite to the Cu-1st cleaner tailings. This would tend to lower the mass of these categories rejected by the Cu-2nd cleaner, and hence back to the Cu-1st cleaner feed. The Cu-1st cleaner tailings is therefore the preferred regrind stream in such a configuration.

Regrinding the Zn-1st cleaner tailings in isolation (i.e. in a dedicated mill within the Zn-circuit), is not considered in terms of increasing chalcopyrite liberation, as recovery to the Cu-concentrate from this point in the Zn-circuit is not feasible. Floatable chalcopyrite will however, report to the Zn-concentrate, and hence be recovered during Cu-smelting of the zinc plant residues. Selection of this stream for regrinding is primarily dedicated to improved sphalerite recovery. The stream contains 8.88 % sphalerite, of which 75.3 % is unliberated. Furthermore, recovery of the constituent middling and sub-middling fractions is 11.1 and 8.43 %, respectively, but accounts for only 2.79 % of the sphalerite in the feed. Thus, regrinding this stream presently would not achieve significant improvement in sphalerite recovery. This stems from the fact that significant amounts of middling and particularly sub-middling grains report directly to the tailings. Thus, regrinding of this stream is only recommended if the sphalerite recovery of the Zn-rougher can be increased (i.e. set to 'pull hard'), and the rejection at the Zn-1st cleaner can be intensified. The mass of the Zn-1st cleaner tailings will thus be increased as will the fractions of middling and sub-middling sphalerite.

Regrinding the Zn-1st cleaner tailings requires (1) installation of a second dedicated regrind mill or (2) directing this stream to the regrind on the Cu-1st cleaner tailings (or Cu-rougher concentrate) after conditioning to desorb the Cu ions adsorbed during Zn-conditioning. Thus, while the use of a single regrind mill dictates conditioning of the Zn-1st cleaner tailings, it is compensated for by improved chalcopyrite recovery to the Cu-concentrate. This stems from the fact that chalcopyrite liberated from middling and sub-middling particles in the Zn-1st cleaner tailings would then be directed to the 'head', and would not be simply recirculated within the Zn-circuit. However, this would probably be offset in part by the proposed regrind of the Cu-1st cleaner tailings, which would minimize these existing chalcopyrite losses to the Zn-circuit.

Excluding the 'recoverable' sphalerite lost to the Cu-concentrate, the principal losses of both chalcopyrite and sphalerite are as middling and particularly sub-middling grains to the tailings. Because of the dominant sub-middling fractions particularly, these grains require extremely fine regrinding to achieve liberation. Thus, a finer primary grind or regrinding the Cu-rougher tailings, necessitates 'wasteful' grinding of coarse-grained and 'hard' gangue, pyrite (much of which is already free) and slag fayalite to a size of 80 % - 26 um. This is not economically feasible, and is technically impractical and unnecessary. In contrast, the optimum regrind strategy should include (1) setting the Cu- and Zn-roughers to 'pull hard' and hence recover the maximum amount of middling and sub-middling grains of chalcopyrite and sphalerite in the respective cleaner concentrates, (2) intensifying rejection of these grains at the respective 1st cleaners, and (3) regrinding these small (in terms of flow rates) cleaner tailing streams, either in separate or in a single regrind mill. It is significant that the grain size of middling plus sub-middling chalcopyrite and sphalerite in the Cu-rougher concentrate and the Cu-1st cleaner tailings are comparable (i.e. approx. 80 % - 40 um; See appendix D). This indicates high predicted liberations if ground to between the minimum grind of 80 % - 40 um (i.e. 50 %) and the optimum grind of 80 % - 18.4 um (i.e. 75 %). The Cu-rougher concentrate (or the Cu-1st cleaner feed) is the ideal stream to regrind except for the potential of over-grinding chalcopyrite. If the latter can be minimized by regrinding a + 50 um hydrocyclone fraction of the Cu-rougher concentrate, it is the preferred stream for regrinding.

The selective regrinding of the Cu-1st cleaner tailings (or the + 50 um fraction of the Cu-rougher concentrate), containing relatively high abundances of both middling chalcopyrite and sphalerite, will tend to improve (1) the minor chalcopyrite losses to the Zn-concentrate as middling chalcopyrite-sphalerite particles; (2) the significant sphalerite losses to the Cu-concentrate as middling chalcopyrite-sphalerite particles; (3) the grade of the Cu-concentrate by reducing the amount of pyrite and gangue (and sphalerite) locked with middling chalcopyrite grains; and (4) the loss of chalcopyrite and sphalerite middling grains to the tailings. In addition to improving the liberation from these 'primary' middling fractions, sub-middling grains may be reground to middling, sub-free or even free grains. These latter composite classes of sphalerite and particularly chalcopyrite (e.g. 92.2 % recovery of middling chalcopyrite in the Cu-concentrate) exhibit only minor losses to the tailings. Thus, shifting the sub-middling grains into more liberated composite classes (without necessarily achieving full liberation) should provide for substantial improvement in chalcopyrite and sphalerite recovery at equivalent concentrate grades, and at the present 'hard pulling' setting of the circuit.

In summary, approximately 10 % of the chalcopyrite and 18 % of the sphalerite in the feed are lost as middling and sub-middling grains from the respective concentrates. These values represent the maximum potential improvement in recovery of the respective mineral, due to the application of selective fine regrinding. However, because of the lack of a single suitable stream for sphalerite particularly, maximum attainable improvement in the recovery of both chalcopyrite and sphalerite would probably approach 10 %, corresponding to improvements of approximately 8 % in Cu and 6 % in Zn recovery.

TABLE 38

PARAMETERS FOR SELECTING REGRIND STREAMS.

PARAMETER.	STREAM. Cu-Roug. Tail.	Cu-Roug. Conc.	Cu-Scav. Conc.	Cu-Cl-1. Tail.	Zn-Cl-1 Tail.
Flow Rate.	95.5	11.3	0.18	6.68	0.88
Recovery of Gangue.	99.2	4.91	0.14	3.93	0.48
Recovery of pyrite.	92.8	25.8	0.30	18.4	2.48
Modal amounts of Gang.+ Py.	96.7	69.2	88.9	88.5	86.2
Cu Assay (%).	0.3	8.4	5.6	2.1	1.2
Cu Recovery (%).	23.4	91.0	0.94	13.5	0.97
Modal Amounts of Chalcopyrite.	0.29	22.7	7.91	6.41	2.13
% Unliberated Cpy.	92.4	70.9	53.1	96.2	91.7
Recovery of middling Cpy.	8.40	147.3	0.59	55.1	0.96
Recovery of sub-middling Cpy.	66.6	108.0	0.71	73.9	4.06
Fraction of Cpy recovery as middling & sub-middling Cpy.	9.61	29.8	0.15	15.6	0.64
Zn Assay (%).	1.5	5.7	5.1	4.5	8.7
Zn Recovery (%).	81.1	36.0	0.49	16.6	4.21
Modal Amounts of Sphalerite.	1.27	4.81	2.39	3.77	8.88
% Unliberated Sphal.	70.5	68.8	35.7	64.9	75.3
Recovery of middling Sphal.	70.2	49.2	0.09	19.2	11.1
Recovery of sub-middling Sphal.	80.1	32.0	0.20	11.8	8.43
Fraction of Sphal recovery as middling & sub-middling Sphal.	22.0	12.3	0.07	4.72	2.79

Note: The Cu and Zn assays and recoveries are biased by Cu and Zn in the slag silicate phases glass and fayalite.

PLATE 2.

(A) Cu-CONCENTRATE : Coarse-grained particle of Intermediate solid solution (Iss) with exsolution lamellae of chalcocite (Cc), and an alteration rim of covellite (Cov). Backscattered electron (BSE).

(B) FEED : Particles of slag (i.e. A to C) showing characteristic texture (e.g. rosette of radiating crystals) of fayalite crystals in the glass matrix. Note sphere of Iss in particle A. Plane Polarised Light (PPL).

(C) TAILINGS : Sub-middling grains of chalcopyrite (Cp) locked in a coarse particle of gangue; characteristic of much of the chalcopyrite lost to the tailings. PPL.

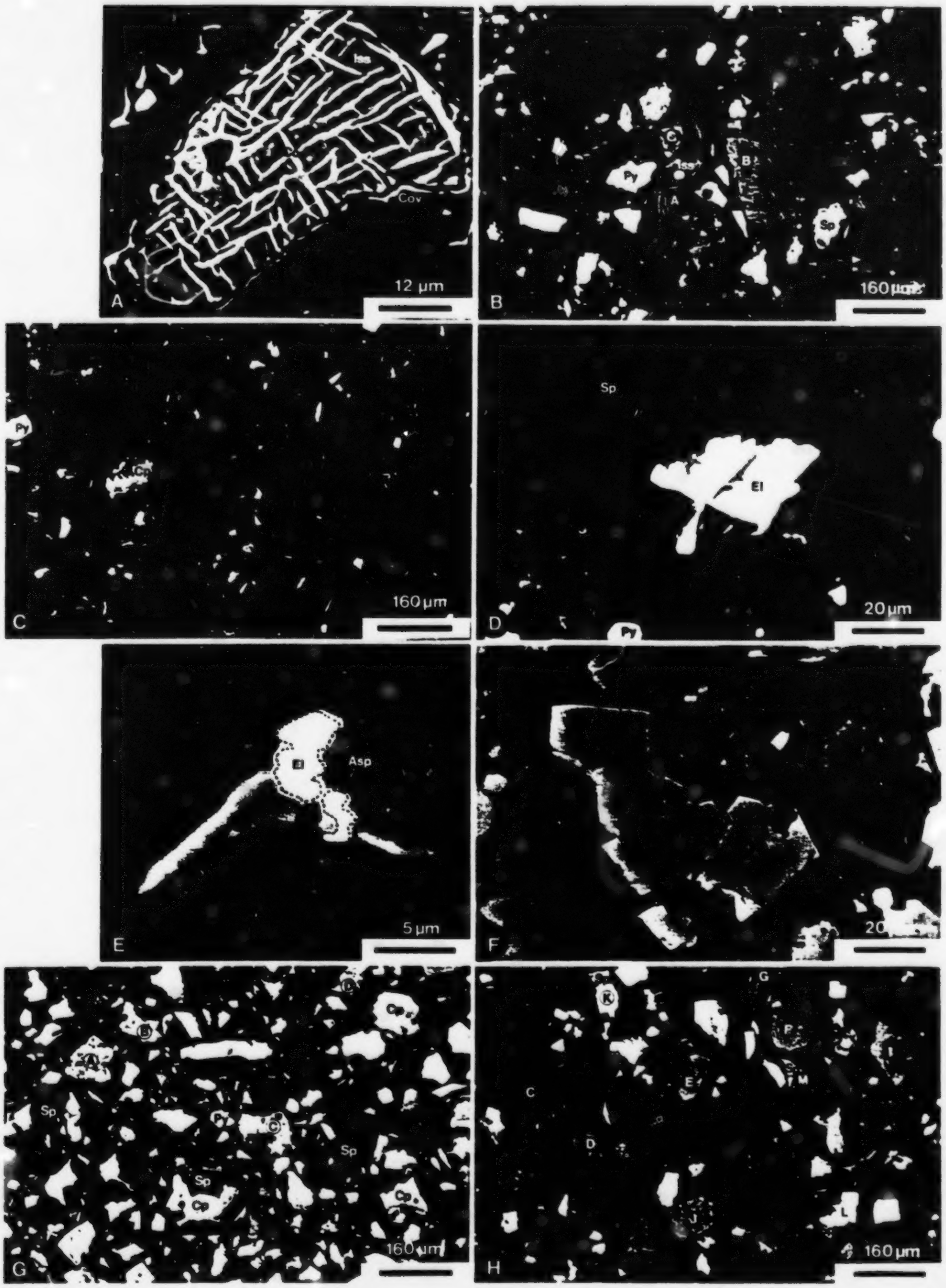
(D) Cu-1st CLEANER CONCENTRATE : Free grain of mercurian electrum (El), with an approximate 25 um 'equivalent square diameter'. Oil Immersion (OI). PPL.

(E) Cu-SCAVENGER TAILINGS : Grain of arsenopyrite (Asp), containing extremely fine-grained inclusions of mercurian electrum, that was rejected from the Cu-circuit. BSE.

(F) Cu-CONCENTRATE : Middling particle of pyrite (Py) and freibergite (Fr) recovered in the Cu-concentrate due to the surface exposure of freibergite. OI. PPL.

(G) Cu-CONCENTRATE : Abundant medium-grained free and sub-free grains of sphalerite, (Sp) and grains of sphalerite locked with pyrite ± chalcopyrite. Note characteristic middling particles of chalcopyrite interstitial to, enveloping, and forming a veneer on pyrite grains (i.e. A to D), and which are amenable to regrinding. PPL.

(H) TAILINGS : Abundant particles of slag (i.e. A to J) which account for approximately 60 % of the Cu and 85 % of the Zn in the tailings. Note the two particles (i.e. K and L) consisting of middling sphalerite-pyrite, and a free grain of sphalerite at M. PPL.


Iss
Cov
12 µm
A
C
Iss
B
Py
A
Sp
160 µm
B
Py
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160 µm
C
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Cp
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Chapter VIII

SIZE ANALYSIS

SIEVE SIZE ANALYSIS OF THE MILL PRODUCTS

Sieve size analysis using Tyler series meshes 48, 65, 100, 150, 200 and 325 were done by HBM&S on the 20 mill product samples. The results are presented in Table 39, and are condensed for the 17 principal mill products in Table 40. It can be seen that feed has the finest particle size distribution of all the samples, with 62.3 % finer than 325 mesh (i.e. 43 um). This contrasts with the Trout Lake feed in which 67.8 % was finer than 325 mesh (Healy & Petruk 1889). Thus, the bulk of the material in the circuit lies below the resolution of the sieving technique. As chalcopyrite and sphalerite tend to be concentrated in the finer sieve fractions of the grind, it is likely that even larger proportions of these minerals of interest are concentrated in the - 325 mesh fraction. It must therefore be stressed that the use of sieve size analysis to evaluate individual mineral behaviour is conjectural in that (1) it does not address the mineralogical nature, but merely the particulate nature of the bulk product, and (2) as approximately 63 % of the material being examined lies below the resolution of the method, the examination is restricted to the coarse tail of the particle size distribution. Nonetheless, sieve size analysis may complement the other analytical techniques employed in this study, though the data presented here, which were not material balanced, do not provide significant new data. Consequently, these data are discussed briefly.

From the data in Table 40, the raw screen size or particle size distributions of selected products are plotted as cumulative size curves in Figures 13 and 14. The cumulative size curves for the four end products (i.e. feed, concentrates and tailings) are plotted in Figure 13A, and show that the Cu-concentrate is the finest of all four products, followed by the Zn-concentrate, tailings and feed. The tailings was expected to be coarser than the feed, unless contrary to the information provided, the Cu-rougher tailings was being reground at the time of sampling. However, the discrepancy probably reflects errors in sampling and analysis of the feed and tailings, which typically are unrepresentative and unreliable samples (W. Petruk, pers. comm. 1988). The finer sieve sizes of the two concentrates indicates that both chalcopyrite and sphalerite are concentrated in the finer sieve fractions of the feed. As significant losses of 'over-sized' chalcopyrite and sphalerite were not apparent from image analysis or petrographic examination, this trend does not reflect differential flotation of finer grained fractions of chalcopyrite and sphalerite into the respective concentrates.

The particle size curves of the feed and concentrates of the Cu-circuit are shown in Figure 13B. The most notable feature of Figure 13B is the fine size of the Cu-scavenger concentrate, and the relatively coarse size of the Cu-rougher concentrate. This reflects the high recovery of middling and possibly 'over-sized' chalcopyrite by the Cu-rougher (i.e. 'hard pulling'), prior to Cu-scavenging. For a given size grain of chalcopyrite, the particle size equals that of the grain when free, whereas when unliberated (and if occurring in a middling particle), the particle size greatly exceeds that of the chalcopyrite grain. These middling and 'over-sized' chalcopyrite grains thus contribute significantly to the coarser particle size distribution of the Cu-rougher concentrate, and conversely for the Cu-scavenger concentrate, which is depleted in the same. Rejection of the Cu-1st cleaner tailings, and re-circulation back of the Cu-2nd cleaner tailings to the Cu-rougher concentrate, results in only a minor drop in the particle size to the Cu-1st cleaner feed. The fine particle size of the Cu-scavenger concentrate is further evidence that it is unsuitable for regrinding, unlike in the Trout Lake circuit, where it was the coarsest product and which Healy & Petruk (1989) recommended for regrinding.

The particle size curves of the feed and concentrates of the Zn-circuit are shown in Figure 14A. The Zn-circuit concentrates are all finer than the Zn-circuit feed, as the latter contains coarse-grained gangue and pyrite that reports principally to the tailings. The Zn-circuit concentrates are progressively coarsened through Zn-cleaning, suggesting rejection of entrained inert fines with minimal rejection of the initial floated particles containing sub-free and middling sphalerite grains, and which therefore tend to be of a coarse size. Furthermore, this is also consistent with coarse particles containing sub-middling sphalerite reporting directly to the tailings, and not being recirculated back by rejection at the cleaners. In the absence of a Zn-scavenger unit, the Zn-rougher tends to 'pull hard' and recover a high proportion of entrained inert fines, which are subsequently rejected by the more selective Zn-1st and 2nd cleaners. However, the 'pull' is insufficient to recover significant amounts of sub-middling sphalerite, which report directly to the tailings.

In Figure 14B, the particle size distributions of the potentially regrindable streams (i.e. the Cu-rougher concentrate and tailings, the Cu-1st and 2nd cleaner tailings, the Cu-scavenger concentrate and the Zn-1st cleaner tailings) are compared. These streams can be placed in three groups on the basis of their respective particle size distributions. The particle size distributions of the Cu-rougher tailings is significantly coarser than those of the other five streams. However, despite being commonly chosen as a regrind, this stream is not suitable for regrinding, for reasons given in the REGRIND section (i.e. principally because it contains 99 % of the gangue and 95 % of the pyrite in the ore). The second group consists of the Cu-rougher concentrate, and Cu-1st and 2nd cleaner tailings. The Cu-2nd cleaner tailings has the finest distribution, and critically, contains the lowest recoveries of middling and sub-middling chalcopyrite and sphalerite of these three streams, and is thus unsuitable for regrinding. The Cu-rougher concentrate and the Cu-1st cleaner tailings have comparable particle size distributions of 77 % - 43 um. These size distributions complement the reasons given in the REGRIND section, recommending these as the two alternative regrind streams; both exhibit high recoveries of middling and sub-middling sphalerite, but particularly chalcopyrite. Group 3 consists of the Cu-scavenger concentrate and the Zn-1st cleaner tailings, which have extremely fine particle size

TABLE 39

RAW SCREEN SIZE ANALYSES FOR THE 20 HBMS MILL PRODUCT SAMPLES.

(Retained Weight i.e. greater than)

SAMPLE.	Mesh Numbers (and Screen Opening Sizes)						
	+48 (295um)	+65 (208um)	+100 (147um)	+150 (104um)	+200 (74um)	+325 (43um)	-325
1. Feed.	1.00	2.92	5.54	7.43	9.33	11.5	62.3
8. Cu-Scav Co.	0.10	0.04	0.14	0.65	2.32	13.4	83.4
10. Cu-Scav Co.	0.03	0.06	0.41	1.22	2.71	15.7	79.8
2. Cu-Roug Co.	0.11	0.07	0.51	1.81	3.60	17.7	76.3
4. Cu-Roug Co.	0.11	0.15	0.56	1.87	3.33	15.5	78.5
6. Cu-Roug Co.	0.09	0.03	0.47	1.73	3.65	17.6	76.4
7. Cu-Cl.1 Fd.	0.05	0.13	0.38	1.62	5.18	12.3	80.3
12. Cu-Cl.1 Co.	0.01	0.07	0.42	1.95	5.81	13.2	78.5
14. Cu-Cl.2 Co.	0.01	0.02	0.44	1.77	3.57	16.3	77.9
15. Zn-Roug Fd.	0.15	0.99	3.48	6.21	9.37	10.8	69.2
16. Zn-Roug Co.	0.05	0.00	0.15	0.93	2.57	16.8	79.5
18. Zn-Cl.1 Co.	0.02	0.01	0.17	1.33	3.66	19.2	75.6
20. Zn-Cl.2 Co.	0.16	0.03	0.24	1.63	4.04	20.9	73.0
3. Cu-Roug Tl.	0.16	1.04	3.21	6.01	9.29	12.6	67.7
5. Cu-Scav Tl.	0.29	1.17	3.58	6.41	9.54	12.8	66.2
9. Cu-Cl.1 Tl.	0.13	0.18	0.51	1.77	3.69	16.3	77.5
11. Cu-Cl.2 Tl.	0.03	0.07	0.41	1.73	5.58	13.4	78.8
19. Zn-Cl.1 Tl.	0.06	0.09	0.21	0.94	3.64	9.70	85.4
21. Zn-Cl.2 Tl.	0.15	0.08	0.16	0.56	2.83	9.35	86.9
17. Zn-Roug Tl.	0.24	0.97	3.29	6.22	9.48	12.4	67.4

TABLE 40

CONDENSED RAW SCREEN SIZE ANALYSES FOR THE 17 HBMS MILL PRODUCTS.

(Retained Weight i.e. greater than)

PRODUCT.	Mesh Numbers (and Screen Opening Sizes)						
	+48 (295um)	+65 (208um)	+100 (147um)	+150 (104um)	+200 (74um)	+325 (43um)	-325
1. Feed.	1.00	2.92	5.54	7.43	9.33	11.5	62.3
2. Cu-Scav Co.	0.07	0.05	0.28	0.94	2.52	14.6	81.6
3. Cu-Roug Co.	0.10	0.08	0.51	1.80	3.53	16.9	77.1
4. Cu-Cl.1 Fd.	0.05	0.13	0.38	1.62	5.18	12.3	80.3
5. Cu-Cl.1 Co.	0.01	0.07	0.42	1.95	5.81	13.2	78.5
6. Cu-Cl.2 Co.	0.01	0.02	0.44	1.77	3.57	16.3	77.9
7. Zn-Roug Fd.	0.15	0.99	3.48	6.21	9.37	10.8	69.2
8. Zn-Roug Co.	0.05	0.00	0.15	0.93	2.57	16.8	79.5
9. Zn-Cl.1 Co.	0.02	0.01	0.17	1.33	3.66	19.2	75.6
10. Zn-Cl.2 Co.	0.16	0.03	0.24	1.63	4.04	20.9	73.0
11. Cu-Roug Tl.	0.16	1.04	3.21	6.01	9.29	12.6	67.7
12. Cu-Scav Tl.	0.29	1.17	3.58	6.41	9.54	12.8	66.2
13. Cu-Cl.1 Tl.	0.13	0.18	0.51	1.77	3.69	16.3	77.5
14. Cu-Cl.2 Tl.	0.03	0.07	0.41	1.73	5.58	13.4	78.8
15. Zn-Cl.1 Tl.	0.06	0.09	0.21	0.94	3.64	9.70	85.4
16. Zn-Cl.2 Tl.	0.15	0.08	0.16	0.56	2.83	9.35	86.9
17. Zn-Roug Tl.	0.24	0.97	3.29	6.22	9.48	12.4	67.4

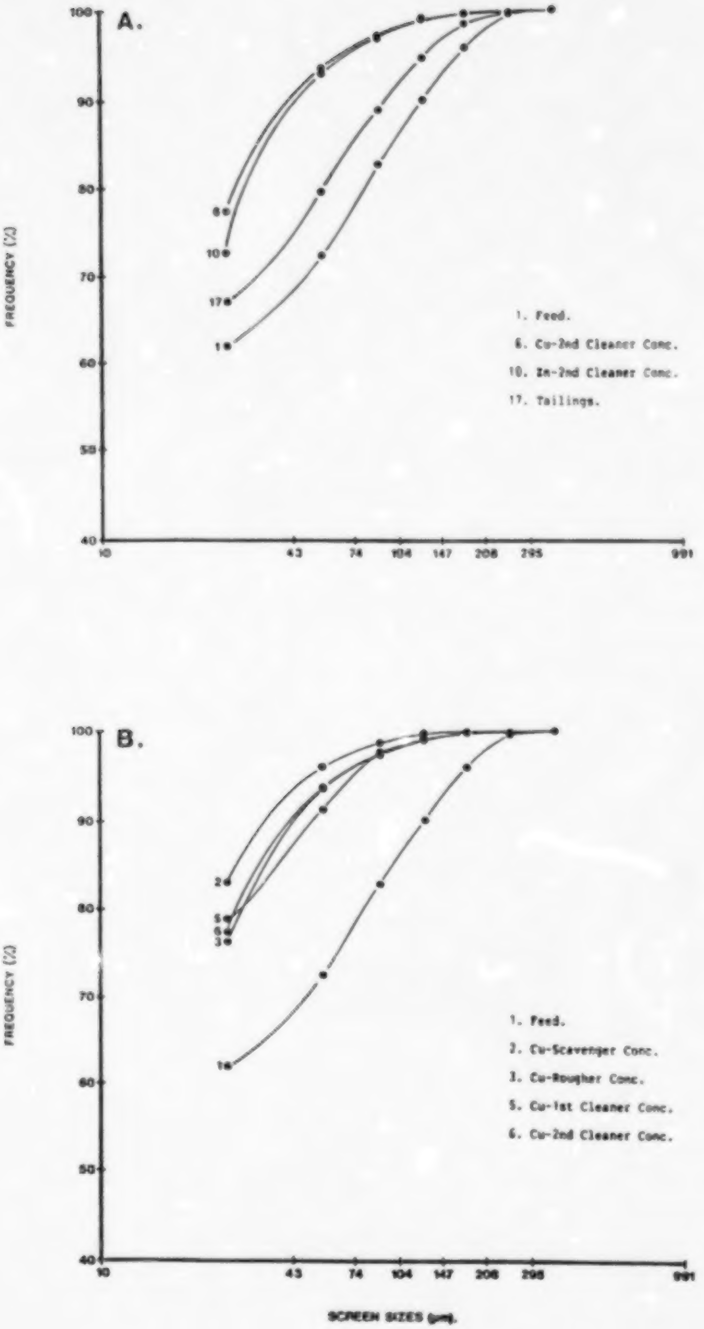

FIGURE 13.
THE PARTICLE SIZE CURVES FOR (A) THE PRINCIPAL END PRODUCTS, AND (B) THE FEED AND CONCENTRATES OF THE Cu-CIRCUIT.

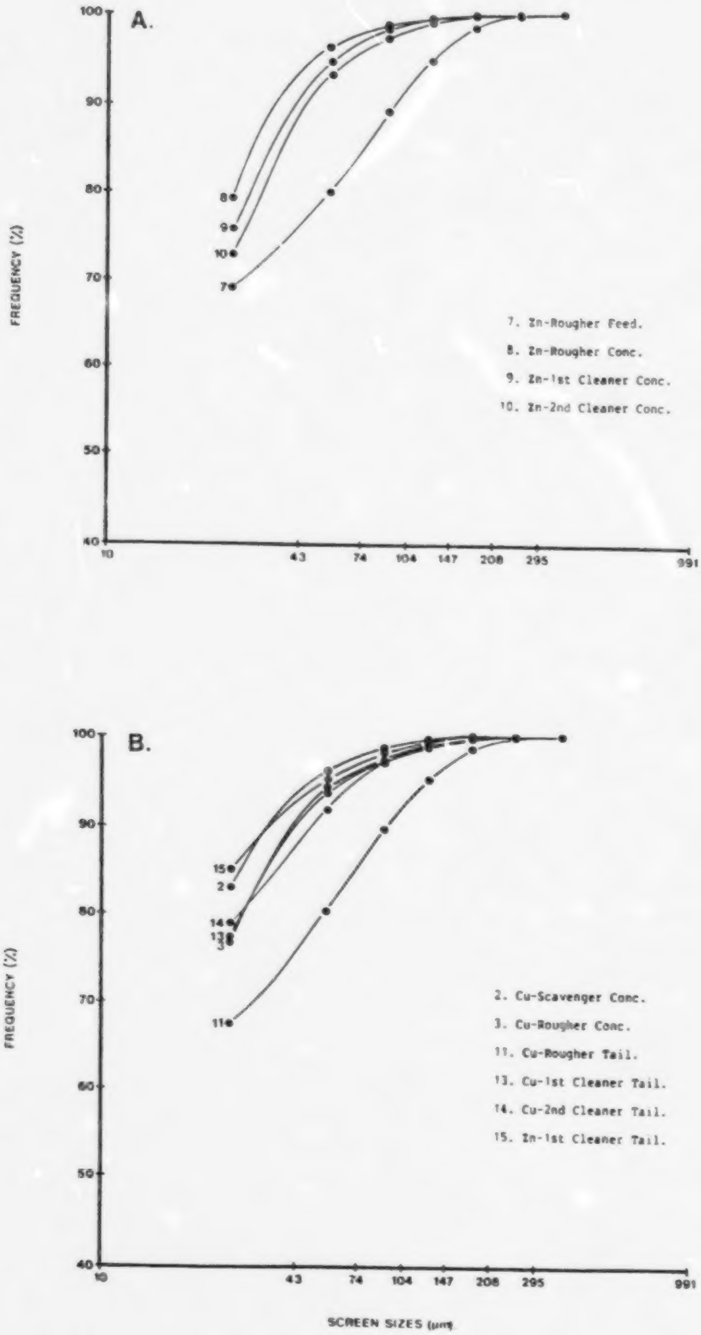

FIGURE 14.
THE PARTICLE SIZE CURVES FOR (A) THE FEED AND CONCENTRATES OF THE Zn-CIRCUIT, AND (B) THE POTENTIALLY REGRINDABLE PRODUCTS.

distributions of 81.6 % and 85.4 %, respectively - 43 um. Unlike in the Trout Lake circuit, the Cu-scavenger concentrate has extremely fine particle sizes, which in addition to the reasons in the REGRIND section, makes it unsuitable for regrinding. The fine particle size distribution of the Zn-1st cleaner tailings makes it a less attractive, though nonetheless on the basis of mineralogical composition regrindable stream (See REGRIND). If the Zn recovery of the Zn-rougher can be increased, many of the coarse particles containing middling and sub-middling sphalerite that presently report directly to the tailings will be recovered, and then rejected to a coarser Zn-1st cleaner tailings. Thus, while the particle size data presented here do not strongly support the recommendations pertaining to a regrind, neither do they negate the latter.

GRAIN SIZES BY IMAGE ANALYSIS

The grain sizes of chalcopyrite, sphalerite and galena were determined using image analysis, by measuring the surface areas of the grains of the respective minerals, and using the square root of the grain area to give an 'equivalent square diameter' (Petruk 1978). As described under PETROGRAPHIC AND IMAGE ANALYSIS, the size distributions of free chalcopyrite and sphalerite grains were determined separately from those of unliberated grains. The grain sizes of free plus unliberated chalcopyrite and sphalerite in each sample were also determined, and from which those of unliberated chalcopyrite and sphalerite were extracted. Before dealing with the grain size distributions of free and of unliberated chalcopyrite and sphalerite in the various products, the grain size of these minerals in the feed, irrespective of their state of liberation, is considered. The data for the feed are presented in Table 41. Also included in Table 41 is the grain size distribution of galena (from page 36 of Volume 2) and of 'all phases' (from Table 1 of appendix C), and the grinding curve determined by sieve size analysis. These distributions are plotted as cumulative curves in Figure 15. It should be stressed that unlike all other grain size data generated by image analysis, the grain size of 'all phases' was determined as the 'equivalent circular diameter'. This gives grain diameters that are greater than the 'equivalent square diameter' for the corresponding grains, and thus particle sizes that are inconsistent with the sieve size analysis.

The grain size distributions of free and unliberated chalcopyrite, and sphalerite in the 17 mill products were not material balanced. The raw grain size distributions for free chalcopyrite and sphalerite in the 20 mill product samples are given in Tables 2 and 3 of appendix C, and are condensed for the 17 principal mill products in Tables 42 and 44, respectively. Similarly, the raw grain size distributions for unliberated chalcopyrite, and sphalerite in the 20 mill product samples are given in Tables 4 and 5 of appendix C, and are condensed for the 17 principal mill products in Tables 43 and 45, respectively. The total surface area of the grains of the minerals of interest, upon which the respective grain size distributions are based, are also given in each of these eight tables. For a given mineral the total surface area is a measure of the statistical uncertainty in the measured value.

From Figure 15 it can be seen that galena, chalcopyrite and sphalerite lie on the finer side of the grind and are thus concentrated in the finer sieve fractions. Consequently, gangue and pyrite which comprise the remainder and bulk of the feed, lie slightly on the coarse side of the grinding curve. The grain size distributions of chalcopyrite, sphalerite and galena are 80 % - 57, - 61 and - 15 um, respectively. With the exception of galena, these contrast with the corresponding distributions in the Trout Lake feed of 80 % - 26, - 36 and - 15 um, respectively (Healy & Petruk 1989). The coarse size of the HBMS grind (i.e. 80 % - 110 um Vs 80 % - 58 um at Trout Lake), and the coarser size of chalcopyrite and sphalerite in the HBMS feed, indicate that chalcopyrite and sphalerite liberations are low in the HBMS feed, as was observed in this study (See MINERAL LIBERATIONS OF CHALCOPYRITE AND SPHALERITE). This assumes comparable grain sizes for these minerals in the unbroken ore fed to the HBMS and Trout Lake circuits. Furthermore, the general relationship of the grain size distributions of galena, chalcopyrite and sphalerite are comparable in both feeds, except for the shift to higher chalcopyrite and sphalerite grain sizes in the HBMS feed. The grain sizes of galena given in Volume 2 are not treated further.

It is interpreted that because of the control the grain size distribution of pyrite exerts on the particle size of the feed or grinding curve (Smith et al 1987), the higher pyrite content of the HBMS feed (i.e. 15.4 % Vs 9.61 % in the Trout Lake feed) accounts for these differences in the grain sizes of chalcopyrite and sphalerite.

TABLE 41

GRAIN SIZE DISTRIBUTIONS OF CHALCOPYRITE, SPHALERITE, GALENA AND THE PARTICLE SIZE CURVES OF THE HBMS FLOAT FEED.

GRAIN DIAMETER (um)	CHALCOPYRITE		SPHALERITE		GALENA		ALL PHASES.	GRINDING CURVE.
	Freq.	Cum.	Freq.	Cum.	Freq.	Cum.	Cum.	Cum.
3.25	0.03	0.03	0.04	0.04	5.46	5.46	0.04	----
4.60	0.39	0.42	0.49	0.53	7.03	12.5	0.18	----
6.50	1.67	2.09	1.69	2.22	14.9	27.4	1.01	----
9.20	3.59	5.68	3.12	5.34	26.4	53.8	2.34	----
13.0	6.21	11.9	3.36	8.70	14.1	68.5	4.34	----
18.5	8.98	20.9	8.84	17.5	18.0	86.5	7.19	----
26.0	12.4	33.3	9.27	26.8	14.2	100.0	11.4	----
36.8	16.2	49.5	15.5	42.3			17.7	----
43.0	----	----	----	----	----	----	----	62.3
52.0	14.5	64.0	15.0	57.3			28.4	----
74.0	22.0	85.9	24.5	81.8			43.5	73.8
104.	14.1	100.0	18.2	100.0			62.7	83.1
147.							81.3	90.5
208.							96.0	96.1
+208.							100.0	100.0

NOTE : Grain size of 'ALL PHASES' (equivalent to particles) determined as the 'equivalent circular diameter'.

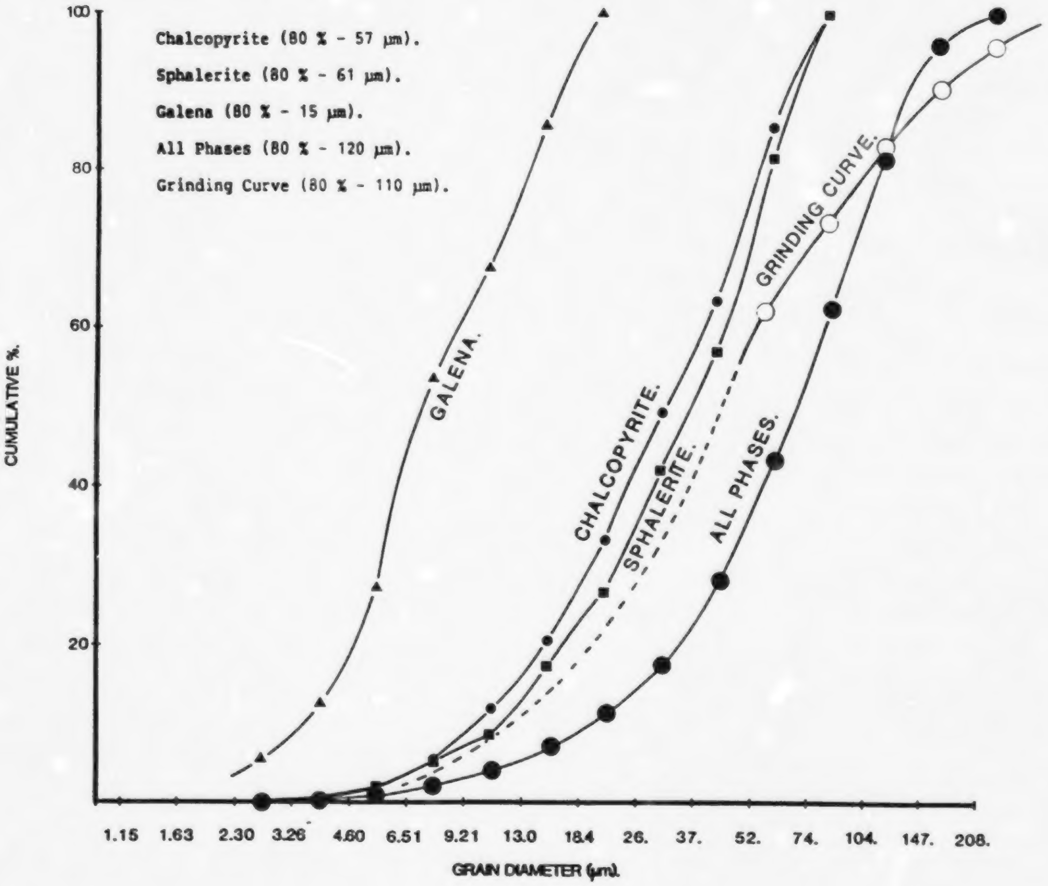

FIGURE 15.
THE GRAIN SIZE CURVES OF CHALCOPYRITE, SPHALERITE AND GALENA, ALL PHASES, AND THE PARTICLE SIZE CURVE OF THE FLOAT FEED.

CHALCOPYRITE

The grain size distributions of free and unliberated chalcopyrite in the 17 mill products are given in Tables 42 and 43, respectively. Plots of the raw grain size distribution of free chalcopyrite and of the grain size distribution and classification of free plus unliberated chalcopyrite in the feed, Cu-concentrate, Zn-concentrate and tailings are given in Figures 1 to 4 of appendix C.

FREE GRAINS

In the chapter on MINERAL LIBERATIONS OF CHALCOPYRITE AND SPHALERITE it was determined by material balance calculation that 97.7 % of the free chalcopyrite is recovered in the Cu-concentrate, suggesting that all grain size classes of free chalcopyrite are essentially completely recovered. The losses (though negligible) calculated by MATBAL reflect (1) apparent free grains of chalcopyrite derived from dominant assemblages of unliberated grains in Zn-circuit products, (2) actual losses of coarse-grained (i.e. over-sized) free chalcopyrite to the Zn-concentrate, and (3) possible losses of extremely fine-grained free chalcopyrite, which are petrographically conspicuous in the Cu-scavenger concentrate, but which were not observed in the tailings. Losses of coarse-grained free chalcopyrite to the Zn-concentrate were observed petrographically to be minor, and much less significant than in the Trout Lake Zn-concentrate (Healy & Petruk 1989). Thus, the discussion of free chalcopyrite, as measured by image analysis is limited to the Cu-circuit products, in which it is predominantly recovered.

The grain size curves for free chalcopyrite in the feed, Cu-circuit concentrates and Cu-1st cleaner tailings are presented in Figure 16A. Four significant observations are apparent. Firstly, the grain size of free chalcopyrite in the feed (i.e. 80 % - 32 um) is comparable to that of the Cu-1st cleaner tailings, but notably finer than that of the Cu-concentrate (i.e. 80 % - 44 um). However, the amount of free chalcopyrite observed in the feed is in excess of an order of magnitude less than that in the Cu-concentrate, and is therefore subject to higher uncertainties. Furthermore, as the maximum grain size of free chalcopyrite in the feed is < 52 um, yet grains up to 147 um were observed in other products, it is apparent that the data for the feed are biased to finer sizes. The raw data suggests that fine-grained free chalcopyrite is lost to the Zn-circuit, though this can be largely explained by relatively minor observations of apparent free grains derived from assemblages of unliberated grains, and which stereologically are preferentially fine-grained.

Secondly, the grain size of free chalcopyrite in the Cu-scavenger concentrate is the finest of all these products, and reflects the high recovery of conspicuous fine-grained free chalcopyrite, and probably apparent free grains derived from the dominant assemblages of medium-grained unliberated chalcopyrite grains. This trend is in contrast with that of the Trout Lake Cu-scavenger concentrate, for which the grain size distribution of free chalcopyrite was the coarsest of all products (Healy & Petruk 1989).

Thirdly, grain size of free chalcopyrite increases from the Cu-rougher concentrate to the the Cu-1st cleaner concentrate, and subsequently decreases to the Cu-2nd cleaner concentrate. This suggests that apparent free grains (which are derived from middling and sub-middling grains and tend to be preferentially fine-grained) are prominent in the Cu-rougher concentrate, but are reduced in the Cu-1st cleaner concentrate, due to relatively high rejection of the latter to the Cu-1st cleaner tailings. In contrast, the decrease in the grain size beyond the Cu-2nd cleaner reflects the preferential rejection of coarse-grained free chalcopyrite to the Cu-2nd cleaner tailings (See Table 42), whose free chalcopyrite exhibits a relatively coarser grain size than that of the Cu-1st cleaner tailings. Thus, the data is consistent with circulating loads consisting preferentially of sub-middling and middling chalcopyrite (which yield observations of apparent fine-grained free chalcopyrite) at the Cu-1st cleaner, and of coarse-grained free chalcopyrite at the Cu-2nd cleaner.

Fourthly, the fine grain size of free chalcopyrite in the Cu-1st cleaner tailings (i.e. 80 % - 30 um) indicates that regrinding this stream will not address the minor losses of coarse-grained free chalcopyrite to the Zn-concentrate. The above data confirm that while free chalcopyrite is largely (i.e. 97.7 %) recovered in the Cu-concentrate, the minor losses are probably concentrated in the extremities of the grain size distribution, as extremely fine-grained (i.e. < 5 or 10 um), or as coarse-grained (i.e. > 100 um) free chalcopyrite. However, the much of the calculated losses are probably spurious, the apparent free grains being derived from unliberated grains in assemblages that consist of generally > 90 % unliberated grains.

UNLIBERATED GRAINS

In the chapter on MINERAL LIBERATIONS OF CHALCOPYRITE AND SPHALERITE it was determined by material balance that 85 % of the unliberated chalcopyrite is recovered in the Cu-concentrate, with losses principally to the tailings (i.e. 13.3 %). Evaluating the behaviour of unliberated grains on the basis of grain size is complicated by the degree of liberation of the grain, and the host particle size; two additional factors that do not influence the behaviour of free grains, and were not considered in evaluating the latter. These effects are considered in evaluating the behaviour of unliberated grains, much in the same manner as stereological derivation of fine-grained apparent free grains from unliberated grains influenced the evaluation of the behaviour of free grains. However, as 68 % of the unliberated chalcopyrite consists of sub-free chalcopyrite, both the liberation of the grain and the size of the host particle, and hence it's flotation behaviour, approach that of an equivalent free grain of chalcopyrite.

The grain size curves of unliberated chalcopyrite in selected products are given in Figure 16B. It is apparent that the grain size distribution of unliberated chalcopyrite in the feed (i.e. 80 % - 60 um) is comparable to that in the Cu-concentrate (i.e. 80 % - 54 um), and that the cumulative curves intersect and cross-over at several points within the size range. However, notably the grain size of unliberated chalcopyrite in the Zn-concentrate (i.e. 80 % - 38 um) and tailings (i.e. 80 % - 39 um) are finer than those of the feed or Cu-concentrate. This indicates that the distribution in the feed is slightly too coarse-grained with respect to that in the Cu-concentrate, or that contrary to the information provided, a regrind was being employed at the time of sampling. Nonetheless, it is apparent that the losses of unliberated chalcopyrite, which are largely to the tailings, are preferentially fine-grained. This is consistent with their occurrence principally as sub-middling grains, and therefore comprising < 30 % of the host particles.

The grain size distribution of unliberated chalcopyrite in the Zn-concentrate is the finest of these products. Contributing to this fine grain size, is (1) chalcopyrite-disease in sphalerite, which typically constitute sub-middling grains of chalcopyrite, and (2) fine-grained diablastic and granular sphalerite-chalcopyrite particles, and (3) pyrite-chalcopyrite and pyrrhotite-chalcopyrite middling particles. The maximum grain size of unliberated chalcopyrite in the Zn-concentrate is less than 74 um, which does not include the rare over-sized (i.e. > 100 um) sub-free grains of chalcopyrite observed petrographically, but which were not measured fortuitously during image analysis.

The grain size distribution of unliberated chalcopyrite in the Cu-scavenger concentrate (i.e. 80 % - 54 um) is finer than that of the feed (i.e. 80 % - 60 um), and this trend contrasts with that of the Trout Lake Cu-scavenger concentrate (i.e. 80 % - 76 um) relative to the Trout Lake feed (i.e. 80 % - 25 um; See Healy & Petruk 1989). This reflects the low recovery of middling and sub-middling chalcopyrite in the Cu-scavenger concentrate, and further confirms that this stream is unsuitable for regrinding.

The grain size distribution of unliberated chalcopyrite in the Cu-rougher concentrate (i.e. 80 % - 74 um) is the coarsest distribution of all these products, and is slightly coarser than that in the Cu-concentrate. This decrease in grain size through Cu-cleaning probably reflects the rejection of 'over-sized' sub-free and middling chalcopyrite, particularly in the Cu-2nd cleaner tailings (See Table 43). The coarse size distribution of the Cu-rougher concentrate further supports the recommendation to regrind it rather than the Cu-1st cleaner tailings, provided over-grinding of already free, medium- to fine-grained chalcopyrite can be avoided.

TABLE 42

GRAIN SIZE DISTRIBUTIONS OF FREE CHALCOPYRITE IN THE 17 HBMS MILL PRODUCTS.

	Product.	Classes (um).													Area. (um²)
		-3.25	-4.60	-6.50	-9.19	-13.0	-18.4	-26.0	-36.8	-52.0	-71.5	-104.	-147.	+147.	
1	Float Feed.	0.00	0.11	2.87	7.14	9.69	24.1	16.5	16.9	22.8	0.00	0.00	0.00	0.00	29800.
2	Cu Scav Co.	0.00	0.94	6.87	14.5	16.7	15.6	24.3	15.2	5.95	0.00	0.00	0.00	0.00	52100.
3	Cu Roug Co.	0.00	0.49	4.11	7.19	9.29	12.0	17.0	13.5	18.9	7.05	7.38	3.03	0.00	323200.
4	Cu Cl 1 Fd.	0.00	0.63	4.35	6.40	8.40	11.7	13.3	22.8	14.7	11.3	4.46	0.00	0.00	114000.
5	Cu Cl 1 Co.	0.00	0.15	1.41	3.02	4.01	7.14	9.72	14.3	21.3	22.8	6.41	7.62	0.00	117000.
6	Cu Cl 2 Co.	0.01	0.22	2.10	4.07	6.49	8.13	13.6	17.1	21.6	20.9	3.90	1.84	0.00	520000.
7	Zn Roug Fd.	0.00	5.20	22.8	72.0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	219.
8	Zn Roug Co.	0.00	0.82	5.62	4.29	2.30	7.43	8.59	9.71	32.5	28.7	0.00	0.00	0.00	8170.
9	Zn Cl 1 Co.	0.19	1.63	5.61	9.02	9.47	3.95	4.04	0.00	66.1	0.00	0.00	0.00	0.00	7510.
10	Zn Cl 2 Co.	0.00	0.11	1.31	2.80	2.13	2.99	2.36	5.05	18.2	0.00	0.00	65.0	0.00	14100.
11	Cu Roug Tl.	0.61	1.14	5.87	0.00	17.0	16.3	0.00	59.0	0.00	0.00	0.00	0.00	0.00	1150.
12	Cu Scav Tl.	0.00	2.97	8.91	21.0	23.0	44.2	0.00	0.00	0.00	0.00	0.00	0.00	0.00	885.
13	Cu Cl 1 Tl.	0.00	0.60	4.53	7.35	13.4	8.74	15.6	32.0	17.7	0.00	0.00	0.00	0.00	13900.
14	Cu Cl 2 Tl.	0.00	1.52	8.55	14.5	12.0	12.0	19.1	0.00	32.4	0.00	0.00	0.00	0.00	14759.
15	Zn Cl 1 Tl.	0.66	2.92	15.0	13.6	10.9	19.0	37.9	0.00	0.00	0.00	0.00	0.00	0.00	2250.
16	Zn Cl 2 Tl.	0.00	0.94	12.0	19.6	19.2	8.12	0.00	40.2	0.00	0.00	0.00	0.00	0.00	4550.
17	Zn Roug Tl.	0.00	3.67	5.18	20.1	21.2	49.9	0.00	0.00	0.00	0.00	0.00	0.00	0.00	406.

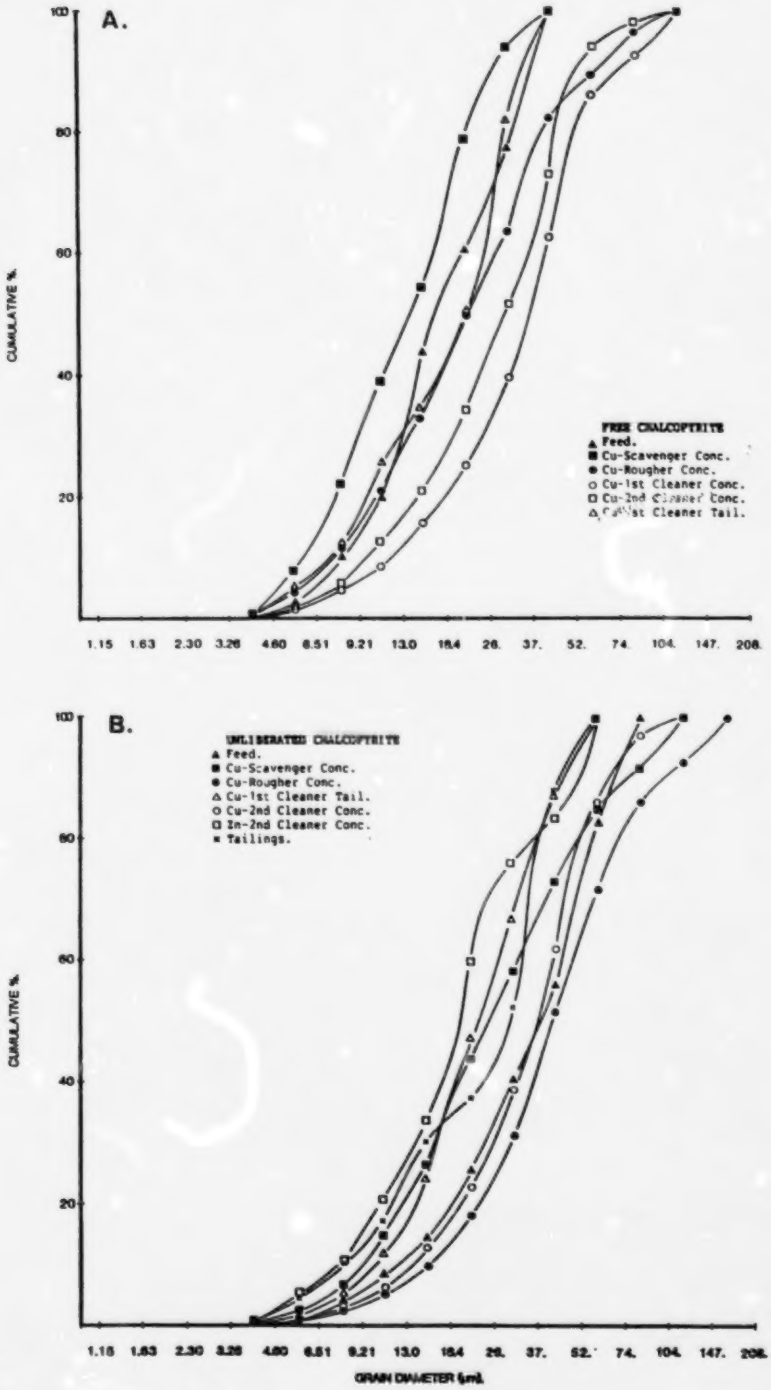

FIGURE 16.
THE GRAIN SIZE CURVES OF (A) FREE, AND (B) UNLIBERATED CHALCOPYRITE IN SELECTED MILL PRODUCTS.

TABLE 43

GRAIN SIZE DISTRIBUTIONS OF UNLIBERATED CHALCOPYRITE IN THE 17 HBMS MILL PRODUCTS.

	Product.	Classes (um).													Area. (um²)
		-3.25	-4.60	-6.50	-9.19	-13.0	-18.4	-26.0	-36.8	-52.0	-73.5	-104.	-147.	+147.	
1	Float Feed.	0.02	0.36	1.20	2.52	4.28	6.25	11.2	14.8	15.6	26.7	17.1	0.00	0.00	160500.
2	Cu Scav Co.	0.02	0.67	1.99	4.03	8.12	11.5	17.6	14.5	14.6	11.9	6.56	8.54	0.00	144100.
3	Cu Roug Co.	0.10	0.18	0.74	1.50	2.67	4.79	8.44	13.1	20.2	20.1	14.4	6.57	7.40	1504600.
4	Cu Cl 1 Fd.	0.01	0.25	0.74	1.55	2.83	4.53	8.64	11.7	15.6	24.1	18.1	7.41	4.59	497000.
5	Cu Cl 1 Co.	0.01	0.06	0.34	0.90	1.68	3.46	4.75	9.61	14.7	18.6	26.4	10.7	8.61	729000.
6	Cu Cl 2 Co.	0.01	0.20	0.96	1.90	3.33	6.40	10.0	16.1	22.9	24.2	11.1	3.12	0.00	413000.
7	Zn Roug Fd.	0.26	4.86	8.48	7.00	15.7	30.5	7.48	25.7	0.00	0.00	0.00	0.00	0.00	6344.
8	Zn Roug Co.	0.10	1.75	5.12	7.79	8.31	9.84	12.8	28.8	10.7	4.73	10.0	0.00	0.00	71350.
9	Zn Cl 1 Co.	0.17	1.65	4.36	5.94	7.25	10.2	10.8	15.8	16.9	26.9	0.00	0.00	0.00	86910.
10	Zn Cl 2 Co.	0.06	1.13	4.31	5.42	9.85	13.2	25.1	17.2	7.05	16.5	0.00	0.00	0.00	52200.
11	Cu Roug Tl.	0.01	1.79	4.50	8.71	15.0	16.2	26.5	18.8	8.42	0.00	0.00	0.00	0.00	16210.
12	Cu Scav Tl.	0.01	1.48	2.65	5.45	10.7	16.0	18.1	14.4	31.2	0.00	0.00	0.00	0.00	25130.
13	Cu Cl 1 Tl.	0.03	0.40	1.76	3.46	6.55	12.0	23.0	19.7	20.2	13.0	0.00	0.00	0.00	196400.
14	Cu Cl 2 Tl.	0.00	0.28	1.22	2.49	5.47	9.08	14.6	18.0	18.6	21.5	2.57	6.15	0.00	241800.
15	Zn Cl 1 Tl.	0.18	2.07	5.60	8.96	9.92	21.4	15.9	25.9	10.1	0.00	0.00	0.00	0.00	42490.
16	Zn Cl 2 Tl.	0.01	2.08	6.93	11.2	14.5	13.9	20.8	19.5	11.2	0.00	0.00	0.00	0.00	18880.
17	Zn Roug Tl.	0.08	0.81	3.64	6.00	6.88	12.8	7.32	14.7	35.4	12.4	0.00	0.00	0.00	21760.

SPHALERITE

The grain size distributions of free and unliberated chalcopyrite in the 17 mill products are given in Tables 44 and 45, respectively. Plots of the raw grain size distribution of free chalcopyrite and of the grain size distribution and classification of free plus unliberated chalcopyrite in the feed, Cu-concentrate, Zn-concentrate and tailings are given in Figures 5 to 8 of appendix C.

FREE GRAINS

In the chapter on MINERAL LIBERATIONS OF CHALCOPYRITE AND SPHALERITE it was detemined by material balance calculation that 77.7 % of the free sphalerite is recovered in the Cu-concentrate, 19.4 % in the Cu-concentrate and 2.88 % in the tailings. These losses of free grains are by definition independent of liberation, and thus grain size possibly exerts the critical control on the loss of these grains.

From the data in Table 44, and the corresponding grain size curves in Figure 17A and B, it is apparent that free sphalerite in the feed (i.e. 80 % - 46 um), is fractionated on the basis of grain size into the Zn-concentrate (i.e. 80 % - 56 um) and the Cu-concentrate (i.e. 80 % - 33 um). Furthermore, it is apparent from Figure 17A that the grain size distribution of all the Cu-circuit concentrates are finer than that of the feed. However, the slime (i.e. < 5 um) fraction constitutes less than 7 % of the sphalerite in each of these products, negating entrainment of sphalerite slimes as a significant contributing mechanism. The grain size of free sphalerite increases from the Cu-rougher concentrate to the Cu-1st cleaner concentrate, and then decreases in the Cu-concentrate, but with a net increase in grain size through the Cu-cleaning. Thus, fine-grained free sphalerite is preferentially recovered in the Cu-concentrate, suggesting an undetermined hydraulic control affecting the seemingly 'non-inert' flotation behaviour of sphalerite prior to activation with $CuSO_4$.

The grain size curves for free sphalerite in the feed and Zn-circuit feed and concentrates are given in Figure 17B. The distribution in the tailings is the finest of all the products, though the slime fraction of this free sphalerite is less than 7 %. It is probable that much of these apparently free sphalerite grains are derived stereologically from unliberated grains, and thus are preferentially fine-grained. Thus, the calculated (by MATBAL) loss of 2.88 % of the free sphalerite to the tailings is taken to be a maximum value. Nonetheless, the trend of increasing grain size of free sphalerite from the Zn-rougher concentrate (i.e. 80 % - 39 um) to the Zn-1st cleaner concentrate (i.e. 80 % - 51 um) and to the Zn-2nd cleaner concentrate (i.e. 80 % - 56 um) also suggests that fine-grained sphalerite is preferentially rejected in the Zn-circuit. However, this trend parallels that of rejection of middling and sub-middling sphalerite; grains which are apt to give abundant observations of fine-grained apparently free grains in the tailings, Zn-1st cleaner tailings (thus the Zn-rougher feed into which it is returned), and the Zn-2nd cleaner tailings. Thus, while the Cu-circuit preferentially recovers fine-grained free sphalerite, the losses to the tailings, also occurring preferentially as fine-grained, free sphalerite, are potentially spurious, being the result of a stereological bias. Furthermore, the loss of free sphalerite to the tailings, even if principally corresponding to actual free grains, is nonetheless within the limits of routine and acceptable losses.

UNLIBERATED GRAINS

In the chapter on MINERAL LIBERATIONS OF CHALCOPYRITE AND SPHALERITE it was determined by material balance that 67.4 % of the unliberated sphalerite is recovered in the Zn-concentrate, 19.9 % in the Cu-concentrate, and 12.7 % in the tailings. As stated earlier, evaluating the behaviour of unliberated grains on the basis of grain size is complicated by the degree of liberation of the grain, and the host particle size. These two factors, which do not influence the behaviour of free grains, may bear heavily on the evaluation of behaviour of unliberated grains.

The grain size curves of unliberated sphalerite in selected Cu-circuit products are given in Figure 18A. It is apparent that, with the exception of the Cu-scavenger concentrate (i.e. 80 % - 25 um), the grain size distribution in the Cu-concentrate (i.e. 80 % - 34 um) is finer than those of other Cu-circuit products and the feed (i.e. 80 % - 63 um). Thus, as with free sphalerite, fine-grained unliberated sphalerite is preferentially recovered in the Cu-concentrate, with a slight fining trend during Cu-cleaning. The finer grain size of unliberated sphalerite in the Cu-concentrate may in part reflect preferential recovery of middling and sub-middling sphalerite locked with chalcopyrite. Middling and sub-middling grains comprise 40 % of the sphalerite in the Cu-concentrate, but only 30 % of that in the feed, and 18 % of that in the Zn-concentrate. Nonetheless, the maximum grain size of unliberated sphalerite in the Cu-concentrate (i.e. < 54 um) is two sieve sizes less than that in the feed (i.e. < 104 um), suggesting that the sub-free sphalerite is also relatively finer-grained with respect to that in the feed. Thus, as with free sphalerite, there is a distinct tendency for fine-grained unliberated sphalerite to be preferentially recovered in the Cu-concentrate. However, this sphalerite contains only a minor (i.e. < 7 %) slime fraction, negating significant entrainment of sphalerite slimes. It is noteworthy, that the grain size of unliberated sphalerite in the Cu-rougher concentrate and the Cu-1st cleaner tailings are comparable (i.e. 80 % - 46 um), and in this regard these streams are thus equally amenable to regrinding.

The grain size distributions of unliberated sphalerite in the feed, Zn-circuit feed and concentrates, and the tailings are given in Figure 18B. With the exception of the distributions in the tailings (i.e. 80 % - 37 um) and Zn-rougher feed (i.e. 80 % - 60 um), which are finer than that of the feed (i.e. 80 % - 62 um), the distributions in the remaining products are either identical or slightly coarser than that of the feed. The finer sizes of the Zn-rougher feed and tailings tends to confirm that the finer-grained unliberated sphalerite occurring as sub-middling grains are preferentially rejected to the tailings. In contrast, coarse-grained unliberated sphalerite (and thus grains more likely to occur as sub-free and middling grains for a given particle size), are essentially fully recovered in the Zn-concentrate. Thus, no losses of 'over-sized' sphalerite are indicated, whereas the losses are preferentially fine-grained and are consistent with sub-middling grains, which comprise the bulk of the sphalerite in the tailings, as shown in MINERAL LIBERATIONS OF CHALCOPYRITE AND SPHALERITE.

TABLE 44

GRAIN SIZE DISTRIBUTIONS OF FREE SPHALERITE IN THE 17 HBMS MILL PRODUCTS.

	Product.	Classes (um).													Area. (um^2)
		-3.25	-4.60	-6.50	-9.19	-13.0	-18.4	-26.0	-36.8	-52.0	-73.5	-104.	-147.	+147.	
1	Float Feed.	0.06	0.42	3.34	5.71	6.46	10.6	15.2	19.6	16.3	22.2	0.00	0.00	0.00	53200.
2	Cu Scav Co.	0.00	1.50	6.31	16.9	14.8	17.8	19.9	4.78	0.00	0.00	0.00	0.00	0.00	36000.
3	Cu Roug Co.	0.01	1.13	6.62	12.2	15.6	17.7	22.2	13.9	5.73	4.99	0.00	0.00	0.00	111400.
4	Cu Cl 1 Fd.	0.02	0.89	7.08	8.90	15.3	19.4	9.47	15.7	9.96	0.00	13.3	0.00	0.00	46500.
5	Cu Cl 1 Co.	0.40	2.38	4.97	7.35	8.73	11.1	16.8	18.3	23.4	6.55	0.00	0.00	0.00	74700.
6	Cu Cl 2 Co.	0.02	0.98	4.92	9.52	13.3	18.3	17.1	13.8	13.2	8.85	0.00	0.00	0.00	89000.
7	Zn Roug Fd.	0.10	1.04	3.20	9.82	13.4	14.4	22.5	12.7	22.9	0.00	0.00	0.00	0.00	14500.
8	Zn Roug Co.	0.02	0.63	3.23	5.84	7.43	11.4	15.2	18.6	25.3	11.4	1.02	0.00	0.00	470000.
9	Zn Cl 1 Co.	0.01	0.43	2.07	3.99	5.97	8.22	12.4	16.0	19.9	23.5	7.59	0.00	0.00	840000.
10	Zn Cl 2 Co.	0.01	0.26	1.47	3.12	4.55	7.25	10.4	16.0	20.3	22.0	11.5	1.91	1.23	1530000.
11	Cu Roug Tl.	0.00	0.78	5.24	8.25	11.8	20.6	16.3	31.4	5.51	0.00	0.00	0.00	0.00	20300.
12	Cu Scav Tl.	0.02	0.25	1.48	5.11	5.68	6.52	8.30	17.6	8.43	14.3	32.3	0.00	0.00	41600.
13	Cu Cl 1 Tl.	0.01	0.36	2.35	4.84	9.42	10.0	12.7	14.6	16.1	20.3	9.29	0.00	0.00	81200.
14	Cu Cl 2 Tl.	0.03	0.55	4.57	7.75	9.40	11.1	18.6	8.23	10.8	15.3	13.7	0.00	0.00	70500.
15	Zn Cl 1 Tl.	0.03	1.94	8.01	11.3	12.1	13.0	19.2	14.2	9.83	10.4	0.00	0.00	0.00	73300.
16	Zn Cl 2 Tl.	0.02	1.64	9.00	13.7	15.7	14.3	15.7	13.0	8.45	8.47	0.00	0.00	0.00	207000.
17	Zn Roug Tl.	0.17	2.95	8.03	10.6	25.1	27.9	25.2	0.00	0.00	0.00	0.00	0.00	0.00	4040.

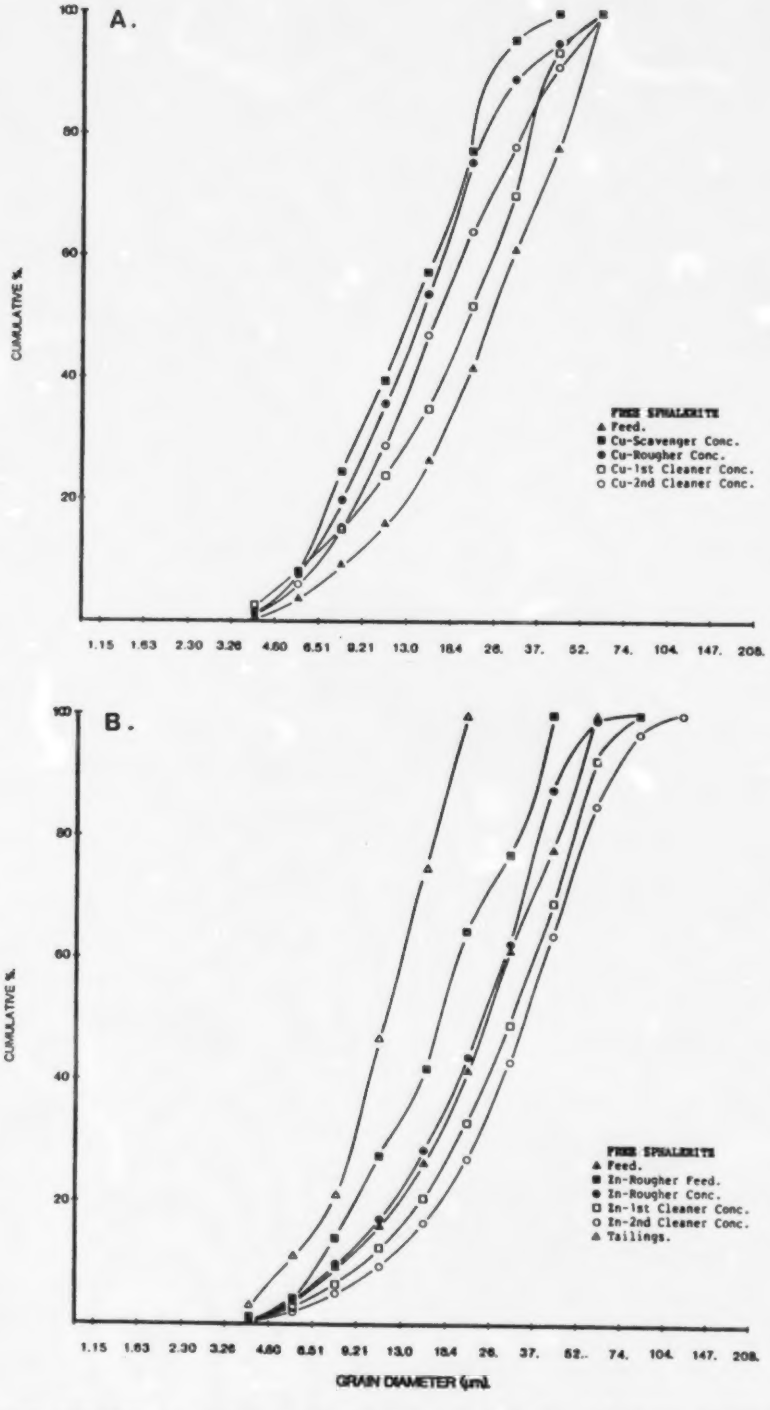

FIGURE 17.
THE GRAIN SIZE CURVES OF FREE SPHALERITE IN (A) Cu-CIRCUIT PRODUCTS, AND (B) Zn-CIRCUIT PRODUCTS.

TABLE 45

GRAIN SIZE DISTRIBUTIONS OF UNLIBERATED SPHALERITE IN THE 17 HBMS MILL PRODUCTS.

	Product.	Classes (um).													Area. (um²)
		-3.25	-4.60	-6.50	-9.19	-13.0	-18.4	-26.0	-36.8	-52.0	-73.5	-104.	-147.	+147.	
1	Float Feed.	0.02	0.29	0.95	2.05	2.59	8.09	6.78	14.5	16.7	26.5	21.6	0.00	0.00	153100.
2	Cu Scav Co.	0.17	1.20	5.32	11.4	12.7	9.44	32.1	19.1	8.65	0.00	0.00	0.00	0.00	71300.
3	Cu Roug Co.	0.04	0.46	1.68	3.56	7.23	10.3	16.8	15.6	21.3	14.5	3.02	5.62	0.00	200800.
4	Cu Cl 1 Fd.	0.00	0.38	1.04	3.41	5.77	8.84	9.65	22.5	20.3	14.3	14.0	0.00	0.00	85600.
5	Cu Cl 1 Co.	0.23	0.71	1.58	2.68	5.32	5.46	12.8	27.7	16.1	7.36	20.2	0.00	0.00	87400.
6	Cu Cl 2 Co.	0.05	0.83	2.66	5.98	7.97	13.1	13.6	27.0	29.0	0.00	0.00	0.00	0.00	55600.
7	Zn Roug Fd.	0.03	0.21	1.02	2.62	4.26	7.32	16.0	9.93	17.5	20.6	20.7	0.00	0.00	39520.
8	Zn Roug Co.	0.01	0.17	0.51	1.28	2.24	4.41	8.01	13.8	21.3	25.0	12.8	10.7	0.00	798000.
9	Zn Cl 1 Co.	0.00	0.12	0.40	0.95	2.05	4.14	8.07	14.3	20.0	28.9	19.7	1.51	0.00	1080000.
10	Zn Cl 2 Co.	0.01	0.08	0.37	0.84	1.90	4.56	7.71	15.2	22.5	23.1	20.7	3.15	0.00	1210000.
11	Cu Roug Tl.	0.00	0.49	1.30	2.64	4.06	11.1	12.5	22.6	20.7	24.7	0.00	0.00	0.00	56000.
12	Cu Scav Tl.	0.05	0.27	1.24	2.15	4.62	7.79	9.54	18.4	10.1	22.4	23.3	0.00	0.00	56100.
13	Cu Cl 1 Tl.	0.02	0.28	1.81	4.03	7.21	10.2	9.74	16.5	29.1	21.0	0.00	0.00	0.00	84400.
14	Cu Cl 2 Tl.	0.00	0.33	1.43	2.44	4.19	6.72	14.7	20.5	15.0	20.1	14.8	0.00	0.00	88900.
15	Zn Cl 1 Tl.	0.04	0.94	3.52	7.51	10.2	15.2	23.3	25.8	10.8	2.57	0.00	0.00	0.00	149700.
16	Zn Cl 2 Tl.	0.03	1.20	4.09	8.85	12.2	17.6	19.2	20.0	13.6	3.28	0.00	0.00	0.00	202000.
17	Zn Roug Tl.	0.08	0.71	1.59	4.69	5.85	12.9	26.9	18.4	16.1	12.8	0.00	0.00	0.00	22830.

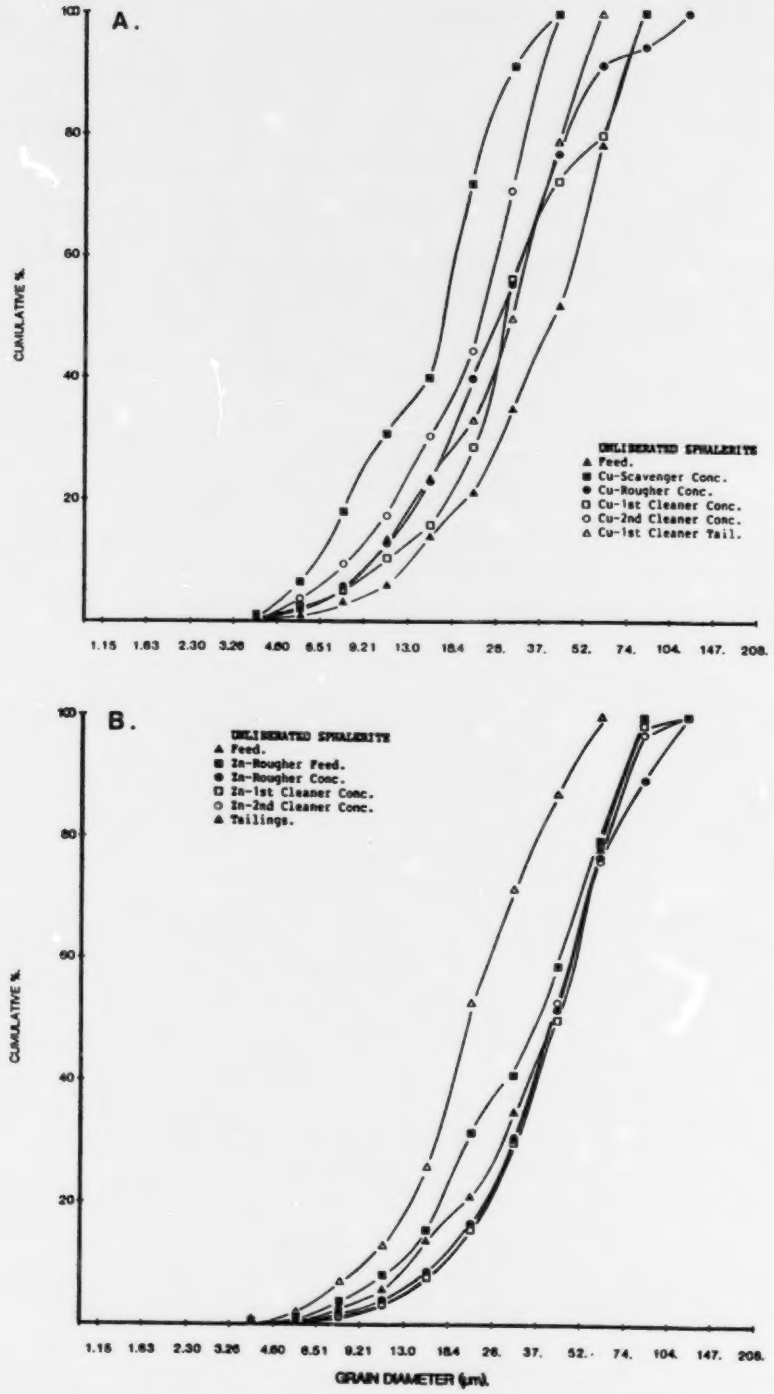

FIGURE 18.
THE GRAIN SIZE CURVES OF UNLIBERATED SPHALERITE IN (A) Cu-CIRCUIT PRODUCTS, AND (B) Zn-CIRCUIT PRODUCTS.

Chapter IX

CONCLUSIONS

1. Hand sorting of the rod mill feed grab sample shows that it contains approximately 34 % barren waste particles, 9.4 % slag, 19 % Chalcopyrite Stringer ore, 18 % Massive Pyrite ore, 18 % Banded Pyrite + Sphalerite ore, and minor amounts of other ore types.

2. The slag consists of Cu- and Zn-bearing silicate phases glass and fayalite, as well as covellite derived from alteration of S-rich species of chalcocite, Iss, bornite, and bornite-chalcocite solid solution. The latter phases are produced by quenching of entrained Cu-matte in the slag. This slag is introduced into the ore by mine-dilution from backfill.

3. The grain size distributions of chalcopyrite, sphalerite and galena in the feed are 80 % - 57 um, 80 % - 61 um and 80 % - 15 um, respectively, while the particle size distribution of the feed (i.e. the grinding curve) is 80 % - 110 um. These grain and particle sizes are coarser than their equivalents in the Trout Lake circuit feed, indicating less effective grinding of this low grade, relatively pyritic ore.

4. Material balance calculated composition of the studied feed in terms of the valuable metals, showed that the feed contained

1.04 % Cu	1.03 ppm Au	0.088 % Pb
1.80 % Zn	12.7 ppm Ag	0.006 % Cd

   The feed contained average levels of Zn, Fe, S and Pb, and relatively low levels of Cu, Au and Ag relative to 'normal' 1983 feeds. However, as slag contains approximately 1.5 % Cu and 6.5 % Zn, the 'effective' or 'recoverable' Cu and Zn head grades (i.e. due to chalcopyrite and sphalerite, respectively) are only approximately 0.9 % Cu and 1.2 % Zn.

5. Material balance modelling of the raw chemical assays for the valuable metals Cu, Zn, Au, Ag, Cd and Pb through the circuit under simulated steady state conditions gives the following recoveries (%) :

	Cu-Conc.	Zn-Conc.	Tailings.
Cu	77.5	1.64	20.8
Zn	19.4	40.2	40.5
Au	49.0	4.90	46.3
Ag	61.3	6.22	32.5
Cd	17.1	31.7	51.2
Pb	17.2	8.95	73.8

6. Unrecoverable Cu and Zn in slag account for approximately 10 % of the Cu, and 35 to 40 % of the Zn in the feed, and approximately 60 % of the Cu and 85 % of the Zn in the tailings.

7. Material balance modelling of the raw chemical assays for the metallurgically troublesome and environmentally hazardous elements Se, Sb, Sn, As and Hg through the circuit under simulated steady state conditions gives the following recoveries (%) :

	Cu-Conc.	Zn-Conc.	Tailings.
Se	15.7	1.99	82.3
Sb	22.5	4.79	72.7
Sn	18.4	0.54	81.1

	Cu-Conc.	Zn-Conc.	Tailings.
As	8.15	3.05	88.8
Hg	21.0	26.6	52.4

8. The distribution of the recoveries of each of the 15 elements indicates four principal groups of elements exhibiting relatively uniform distribution in the products. These are

   a) Group 1 : Report principally to the tailings i.e. Fe, Co, S and As.

   b) Group 2 : Report largely to the tailings, with moderate recoveries in the Cu-concentrate and minor recoveries in the Zn-concentrate i.e. Sb, Sn, Se and Pb.

   c) Group 3 : Exhibit high initial recoveries to the Cu-rougher concentrate (i.e. > 65 %), but with between 40 to 80 % reporting to the Cu-concentrate, low recoveries to the Zn-concentrate (i.e. < 10 %) and moderate recoveries in the tailings i.e. Au, Ag, Sn and In.

   d) Group 4 : Report in minor amounts to the Cu-concentrate, and with comparable and moderately high recoveries (i.e. between 25 and 55 %) to the Zn-concentrate and tailings i.e. Zn, Cd and Hg.

9. The mineralogical composition (in vol %) of the studied feed was calculated by material balances (of image analysis data) to be

78.79 % Gangue (i.e. silicates and carbonates)	
15.35 % Pyrite (including minor magnetite and marcasite)	
2.41 % Chalcopyrite	
1.50 % Sphalerite	0.032 % Galena
0.18 % Arsenopyrite	0.0004 % Gudmundite
0.15 % Pyrrhotite	0.0001 % Freibergite

The quantities of the trace minerals mercurian electrum, hessite, pyrargyrite, cassiterite, bournonite, rucklidgeite, clausthalite and guanajuatite were not determined, or contained such poor counting statistics that material balance modelling of these through the circuit was not done.

10. The distribution of the recoveries of each of the 9 minerals whose modal quantities were determined, and which were material balance modelled through the circuit, indicate four principal groups of minerals, each with relatively uniform flotation behaviour. These are

    a) Group 1 : Inert minerals which report principally to the tailings i.e. gangue, pyrite and gudmundite.

    b) Group 2 : Minerals with high recoveries (i.e. 65 to 90 %) in the tailings and low recoveries (i.e 5 to 20 %) in the Cu- and Zn-concentrates i.e. arsenopyrite, pyrrhotite and galena.

    c) Group 3 : Minerals that report principally to the Cu-concentrate i.e. chalcopyrite and freibergite.

    d) Group 4 : Minerals principally recovered in the Zn-concentrate i.e. sphalerite.

11. The simplified classification of chalcopyrite in the feed, in terms of the principal composite classes, is 27.2 % free, 49.3 % sub-free, 10.1 % middling and 13.3 % sub-middling grains. This is shown in the table below, along with the recoveries of each composite class in the three end products.

CHALCOPYRITE	FREE	SUB-FREE	MIDDLING	SUB-MIDDLING	TOTAL	Cu-Rec.
LIBERATION in the float feed.	27.2	49.3	10.1	13.3	——	100.0
RECOVERY in the Cu-Conc.	97.7	98.7	92.2	34.1	89.2	77.5
RECOVERY in the Zn-Conc.	0.63	0.27	4.64	4.89	1.45	1.64
RECOVERY in the Tailings.	1.64	1.06	3.18	61.0	9.40	20.8

12. While 27.2 % of the chalcopyrite in the feed occurs as free grains (i.e. % liberation), 76.5 % of the chalcopyrite occurs as free plus sub-free grains. This relatively low fraction of 'recoverable' grains of chalcopyrite implies low 'effective liberation'; expected to behave as free grains. Of this, 97.7 % of the free grains and 98.7 % of the sub-free grains are recovered in the Cu-concentrate, and is largely responsible for the overall recovery of 89.2 % of the chalcopyrite (but only 77.5 % of the Cu) in the feed to the Cu-concentrate. The remainder of the recoverable grains of chalcopyrite is recovered in the Zn-concentrate (i.e. 0.63 % of the free and 0.27 % of the sub-free) and the tailings (i.e. 1.64 % of the free and 1.06 % of the sub-free). The relatively minimal losses of approximately 1.5 % of these recoverable grains correspond to 1.2 % of the chalcopyrite in the feed, and account for approximately 10 % of the chalcopyrite in the tailings. Losses of recoverable grains of this magnitude are within the limits of routine and acceptable losses.

13. In addition to the 76.5 % recoverable chalcopyrite in the feed, 10.1 % occurs as middling grains and 13.3 % as sub-middling grains. The middling grains are largely recovered in the Cu-concentrate (i.e. 92.2 %), behaving essentially as free grains in the 'hard-pulling' circuit. Losses of middling grains (in almost equivalent amounts) to the Zn-concentrate and tailings account for less than 1 % of the chalcopyrite in the feed. As these middling grains are the most amenable to regrinding, but are largely recovered in the Cu-concentrate, regrinding can only be effectively directed at the large sub-middling fraction of chalcopyrite.

14. The sub-middling grains comprise 13.3 % of the chalcopyrite in the feed and are largely recovered in the tailings (i.e. 61.0 %), with only 34.1 % reporting to the Cu-concentrate. These grains lost to the tailings corresond to 8.3 % of the chalcopyrite in the feed and account for approximately 85 % of the chalcopyrite in the tailings. The loss of 4.89 % of the sub-middling grains to the Zn-concentrate corresponding to approximately 0.7 % of the chalcopyrite in the feed, is due in part to chalcopyrite-disease in sphalerite. The loss of 66 % of the sub-middling chalcopyrite grains to the Zn-concentrate and particularly the tailings, corresponding to 9 % of the chalcopyrite and 8 % of the Cu in the feed, is the largest fraction of lost chalcopyrite. These losses are only redressable by extremely fine regrinding of streams preferentially enriched in the sub-middling fraction.

15. Extremely fine regrinding (i.e. 80 % - 26 um) of the small Cu-1st cleaner tailings (i.e. stream flow rate = 6.68) is recommended to increase liberation from the large sub-middling and minor middling fractions of chalcopyrite. Such a regrind would also tend to increase the Cu-concentrate grade. The grain size of unliberated chalcopyrite in the Cu-1st cleaner tailings is however relatively fine-grained (i.e. 80 % - 39 um). The Cu-rougher concentrate is, with one notable exception, ideally suited for regrinding, being highly enriched in these fractions of chalcopyrite, and having the coarsest grain size for unliberated chalcopyrite (i.e. 80 % - 74 um) of the Cu-circuit products. However, it is recommended, only if the potential for 'over-grinding' recoverable chalcopyrite can be minimized.

16. The simplified classification of sphalerite in the feed, in terms of the composite classes, is 28.6 % free, 40.8 % sub-free, 16.3 % middling and 14.3 % sub-middling grains. This is shown in the table below, along with the recoveries of each composite class in the three end products. Values in parenthesis show the recovery in the Zn-concentrate of each parameter as a function of that in the Zn-circuit feed (as opposed to the float feed).

SPHALERITE	FREE	SUB-FREE	MIDDLING	SUB-MIDDLING	TOTAL	Zn Rec.
LIBERATION in the float feed.	28.6	40.8	16.3	14.3	----	100.0
RECOVERY in the Cu-Conc.	19.4	14.6	29.9	20.2	19.5	19.4
RECOVERY in the Zn-Conc.	77.7 (96.5)	83.1 (97.2)	55.4 (79.1)	26.7 (33.5)	71.0 (88.2)	40.2 (49.5)
RECOVERY in the Tailings.	2.88	2.35	14.6	53.1	9.50	40.5

17. While only 28.6 % of the sphalerite in the feed occurs as free grains (i.e. % liberation), 69.4 % of the sphalerite occurs as free plus sub-free grains. This relatively low fraction of 'recoverable' grains of sphalerite indicates a poor 'effective liberation'. While 77.7 % and 19.4 % of the free grains report to the Zn-concentrate and Cu-concentrates, respectively, 83.1 % and 14.6 % of the sub-free grains report to these same concentrates, respectively. The tailings recover 2.88 % of the free, and 2.35 % of the sub-free grains of sphalerite. In all, approximately 97 % of the recoverable sphalerite entering the Zn-circuit are recovered in the Zn-concentrate, indicating that Cu-activation by $CuSO_4$ and flotation by Minerec in the Zn-circuit is operating efficiently. Because the recoverable grains of sphalerite constitute only 69.4 % of the sphalerite in the feed, the minimal loss of approximately 2.5 % of these to the tailings results in a corresponding loss of approximately 2 % of the sphalerite in the feed, and accounts for only approximately 15 % of the sphalerite in the tailings.

18. In addition to the 69.4 % recoverable sphalerite in the feed, 16.3 % occurs as middling grains. These report principally to the Zn-concentrate (i.e. 55.4 %), and correspond to 8.84 % of the sphalerite in the feed. The Cu-concentrate recovers 29.9 % of the middling sphalerite, corresponding to 4.77 % of the sphalerite in the feed, and accounting for approximately 25 % of the sphalerite in the Cu-

concentrate. The tailings recovers 14.6 % of the middling sphalerite, which corresponds to 2.6 % of the sphalerite in the feed, and accounts for approximately 20 % of the sphalerite in the tailings. As 16.3 % of the sphalerite in the feed occurs as middling grains, which is the principal category of particle amenable to regrinding, and of which 45 % is lost to the Cu-concentrate and tailings combined, enhanced liberation and recovery (from middling grains of sphalerite) by regrinding is nominally expected to be relatively significant (i.e. 5 %).

19. Of the 14.3 % of the sphalerite in the feed that occurs as sub-middling grains, only 3.73 % (i.e. 27 %) is recovered in the Zn-concentrate, 2.82 % (i.e. 20 %) is recovered in the Cu-concentrate, and 7.76 % (i.e. 53 %) is recovered in the tailings. The losses of sub-middling sphalerite to the Cu-concentrate constitute 15 % of the sphalerite in the Cu-concentrate, whereas those lost to the tailings constitute approximately 65 % of the sphalerite in the tailings. These combined losses of approximately 10.5 % of the sphalerite in the feed, corresponding to 73 % of the sub-middling sphalerite, are essentially unrecoverable without recourse to extremely fine regrinding.

20. While 19.4 % of the sphalerite in the feed is recovered in the Cu-concentrate, the classification of this in terms of the composite classes of the constituent sphalerite is 5.53 % free, 5.94 % sub-free, 4.77 % middling and 2.82 % sub-middling grains, while 0.34 % remained unaccounted for on recombining the material balances of these. Thus, of the sphalerite in the Cu-concentrate, 29 %, 31 %, 25 % and 15 % occur as free, sub-free, middling and sub-middling grains of sphalerite, respectively. The combined recovery in the Cu-concentrate of recoverable grains of sphalerite is approximately 17 %, which corresponds to 11.5 % of the sphalerite in the feed, and approximately 60 % of the sphalerite in the Cu-concentrate. Thus, 60 % of the sphalerite lost to the Cu-concentrate is recoverable (in the Zn-concentrate), confirmimg that the sphalerite losses are only in part due to poor liberation, but are largely due to poor rejection of sphalerite in the Cu-circuit.

21. The concentration factors for sphalerite through Cu-cleaning increase from 1.97 to 4.31, whereas those of pyrite increase from 1.40 to 1.65, and those of gangue decrease slightly from 0.27 to 0.22. In addition, while 19.5 % of the sphalerite, only 0.99 % of the gangue and 7.48 % of the pyrite in the feed are recovered in the Cu-concentrate. The proportionally lower recoveries of pyrite and gangue indicate that the Cu-concentrate is relatively 'clean', despite the high pyrite content (i.e. 24.7 %). The latter merely reflects the pyritic nature of the ore (i.e. feed contains 15.4 % pyrite). The relatively high rejection of pyrite and particularly gangue from the Cu-circuit therefore indicates that the apparent lack of self-draining of the air bubbles in the froth, due possibly to the frothing properties of the Minerec collector, is not the cause of the sphalerite losses to the Cu-concentrate.

22. The abundance of the activator minerals covellite and chalcocite in the float feed (i.e approximately 0.01 Wt %) is potentially sufficient to cause sphalerite activation. Covellite and chalcocite are introduced into the feed from extraneous slag used as mine backfill, and which comprises part of the mine dilution. While the introduction of the slag and associated covellite and chalcocite into the feed is strongly discouraged, the relatively low modal abundances of covellite and the lack of response of sphalerite flotation in cyanidation tests, militate against covellite and chalcocite activating sphalerite.

23. The grain size distributions of free and unliberated sphalerite in the Zn-concentrate are 80 % - 56 um and 80 % - 58 um, respectively, whereas those of free and unliberated sphalerite in the Cu-concentrate are 80 % - 33 um and 80 % - 34 um, resepectively. Thus, while free and unliberated sphalerite in the Cu-concentrate are finer-grained than those in the Zn-concentrate, the former contain a relatively minor (i.e. 3 %) slime fraction; potentially recovered by mechanical entrainment of sphalerite slimes in the Cu-circuit froths. It is concluded that the entrainment of sphalerite slimes contributes negligibly to the sphalerite losses in the Cu-concentrate, but that the principal causative mechanism of these losses preferentially recovers slightly finer fractions of both free and unliberated sphalerite.

24. It is concluded that poor sphalerite liberation, entrainment of sphalerite slimes, activation and poor self-draining of the air bubbles in the froth, are not the principal mechanism causing the high sphalerite losses to the Cu-concentrate. It is further concluded (as in the Trout Lake circuit) that the Minerec collector lacks the selectivity for efficient chalcopyrite-sphalerite separation. Hence, sphalerite is marginally floatable in the Cu-circuit prior to activation with $CuSO_4$. Because of the poorer chalcopyrite liberation in the HBMS circuit, and the consequent need to have the circuit 'pulling hard', the losses of sphalerite to the Cu-concentrate (which are also observed in the Trout Lake circuit) are exaggerated in the HBMS circuit.

25. Diablastic-textured particles of chalcopyrite-sphalerite comprise approximately 5 % of the unliberated sphalerite in the Cu-concentrate. In addition, minor diablastic sphalerite-chalcopyrite and sphalerite-pyrrhotite particles were observed in the Zn-concentrate. This texture is derived principally from Sheared Chalcopyrite + Sphalerite ores. As in the Trout Lake circuit, liberation from this texture is probably low. However, because of the apparently lower abundances of this ore type (relative to that in the Trout Lake circuit), losses due to this texture of either Cu or Zn to the opposing concentrate are not expected to be significant.

26. Excluding the 'recoverable' sphalerite lost to the Cu-concentrate, (which is not amenable to regrinding) the principal losses of both chalcopyrite and sphalerite are as middling and particularly sub-middling grains to the tailings. Because of the dominant sub-middling fractions particularly, these grains require extremely fine regrinding (i.e. 80 % - 26 um) to achieve liberation. Thus, a finer primary grind

or regrinding the Cu-rougher tailings, necessitates 'wasteful' grinding of coarse-grained and 'hard' gangue, pyrite and slag fayalite, and significant over-grinding of already recoverable chalcopyrite and/or sphalerite. It is recommended that regrinding should be coupled with (1) setting the Cu- and Zn-roughers to 'pull hard' and hence recover the maximum amount of middling and sub-middling grains of chalcopyrite and sphalerite in the respective rougher concentrates, (2) intensifying rejection of these grains at the respective 1st cleaners, and (3) regrinding these small cleaner tailings streams, either in separate or in a single regrind mill (provided the Zn-1st cleaner tailings can be desorbed of Cu ions in the latter case). The Cu-rougher concentrate (or the Cu-1st cleaner feed) are the ideal streams to regrind, except for the potential of over-grinding chalcopyrite. If the latter is proven to be negligible or can be minimized, this stream is the preferred stream for regrinding. It is highly improbable that the maximum potential improvement in sphalerite recovery of 18 % (or 11 % Zn) can be realized, because of the lack of a single suitable stream with high recoveries of middling and sub-middling sphalerite. It is predicted that 10 % improvement in the recoveries of chalcopyrite and sphalerite may be realized, if extremely fine regrinding can be directed at these streams. This translates into possible improved Cu and Zn recoveries of up to approximately 8 and 6 %, respectively.

27. The scarcity of mercurian electrum, pyrargyrite and hessite, and the consequent inability to determine the mineral quantities of these minerals in the circuit products prohibits any meaningful mineral-metal accounting for Au and Ag. The low Au assay (i.e. 1.029 ppm), the high Au losses to the tailings (i.e. 46.3 %), the high pyrite and arsenopyrite content of the ore (i.e. 15.4 and 0.18 %, respectively), and the low modal quantities of electrum in all products, suggests that the 'invisible' Au content of the ore is high; compared to the Trout Lake ores. Nonetheless, in addition to the losses as 'invisible' Au, significant Au is probably lost to the tailings as electrum locked in pyrite and arsenopyrite, due to the poor grind and low liberation. Based on Ag assays and the recoveries of sphalerite in the circuit products, Ag in solid solution in sphalerite can only account for a maximum of 8.5 % of the Ag in the feed. In contrast, it is estimated that freibergite accounts for approximately 27 % of the Ag in the feed. It is further inferred that freibergite, chalcopyrite, pyrite, sphalerite, electrum and galena are the principal Ag-carriers in the ore.

28. Hg in solid solution in sphalerite accounts for a maximum of 40 % of the Hg in the feed, while slag fayalite probably contributes significantly to the Hg head grade. However, the Hg content of the HBMS feed is only one fifth that of Trout Lake (i.e. 4 versus 19 ppm). It is inferred that sphalerite, pyrite and slag fayalite are the principal Hg-carriers in the feed.

29. This study is a good example of where conventional chemical assays and sieve sizing tests, both of which are bulk analytical techniques, do not provide meaningful data on the behaviour of the valuable minerals and their contained metals. Because of the masking effect of the

extraneous Cu- and Zn-bearing slag component, chemical assays can only indicate exceptionally poor Cu and Zn metallurgy, but cannot address the extent of the chalcopyrite or sphalerite losses, nor the possible causative mechanisms. The behaviour of recoverable Cu and Zn occurring as chalcopyrite and sphalerite, respectively, can only be evaluated by rigorous mineralogical analysis, specifically image analysis and material balance modelling of these data.

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Appendix A

TABLE 1

RAW MINERAL LIBERATIONS (% FREE) IN THE 20 HBMS MILL PRODUCT SAMPLES.

SAMPLE.	CHALCOPYRITE	SPHALERITE
HB.MP-1	17.7	15.9
HB.MP-8	54.0	29.8
HB.MP-10	23.2	56.0
HB.MP-2	12.2	19.3
HB.MP-4	29.8	42.9
HB.MP-6	22.2	36.6
HB.MP-7	20.5	31.0
HB.MP-12	33.2	43.6
HB.MP-14	56.4	60.2
HB.MP-15	5.04	20.2
HB.MP-16	12.1	35.2
HB.MP-18	8.91	42.0
HB.MP-20	22.0	53.7
HB.MP-3	7.36	16.3
HB.MP-5	5.29	32.2
HB.MP-9	7.37	45.9
HB.MP-11	8.07	42.7
HB.MP-19	6.82	24.8
HB.MP-21	20.0	46.0
HB.MP-17	3.29	5.64

TABLE 2

RAW % UNLIBERATED MINERAL IN THE 20 HBMS MILL PRODUCT SAMPLES.

SAMPLE.	CHALCOPYRITE	SPHALERITE
HB.MP-1	82.3	84.1
HB.MP-8	46.0	30.2
HB.MP-10	76.8	44.0
HB.MP-2	87.8	80.7
HB.MP-4	70.2	57.1
HB.MP-6	77.8	63.4
HB.MP-7	79.5	69.0
HB.MP-12	66.8	56.4
HB.MP-14	43.6	39.8
HB.MP-15	95.0	79.8
HB.MP-16	87.9	64.8
HB.MP-18	91.1	58.0
HB.MP-20	78.0	46.3
HB.MP-3	92.6	83.7
HB.MP-5	94.7	67.8
HB.MP-9	92.6	54.1
HB.MP-11	91.9	57.3
HB.MP-19	93.2	75.2
HB.MP-21	80.0	54.0
HB.MP-17	96.7	94.4

TABLE 3

COMBINED TOTALS OF THE ADJUSTED % FREE AND % UNLIBERATED MINERAL IN THE 17 HBMS MILL PRODUCTS.

(Combined respective columns from Tables 18 and 23).

PRODUCT.	CHALCOPYRITE	SPHALERITE
1. Feed.	93.8	98.8
2. Cu-Scav Co.	82.2	102.4
3. Cu-Roug Co.	94.7	99.2
4. Cu-Cl.1 Fd.	94.9	99.1
5. Cu-Cl.1 Co.	93.2	99.9
6. Cu.Cl.2 Co.	93.0	100.0
7. Zn-Roug Fd.	100.6	98.6
8. Zn-Roug Co.	93.5	97.9
9. Zn-Cl.1 Co.	91.7	97.9
10. Zn-Cl.2 Co.	90.5	97.8
11. Cu-Roug Tl.	98.8	98.5
12. Cu-Scav Tl.	100.7	98.5
13. Cu-Cl.1 Tl.	103.5	98.3
14. Cu-Cl.2 Tl.	100.0	99.2
15. Zn-Cl.1 Tl.	98.9	100.3
16. Zn-Cl.2 Tl.	99.8	100.4
17. Zn-Roug Tl.	102.2	103.7

TABLE 4

RAW LIBERATION AND CLASSIFICATION DATA FOR CHALCOPYRITE IN THE 20 HBMS MILL PRODUCT SAMPLES.

PRODUCT.	Cu ASSAY.	% LOCKED (0-99.9)	CLASSIFICATION OF UNLIBERATED GRAINS.										% FREE (99.9-100)
			0-10	10-20	20-30	30-40	40-50	50-60	60-70	70-80	80-90	90-99	
Feed 1.	1.19	82.3	10.4	6.60	7.69	1.34	3.22	1.40	3.22	7.40	15.5	25.5	17.7
Cu-Scav Co. 8.	5.70	46.0	4.97	5.04	4.30	3.24	1.18	3.42	1.99	1.32	3.48	17.1	54.0
Cu-Scav Co. 10.	5.40	76.8	6.09	7.64	8.30	2.81	3.55	2.60	4.03	4.33	4.15	33.3	23.2
Cu-Roug Co. 2.	8.40	97.7	3.97	4.28	3.92	4.44	2.33	3.63	4.48	3.90	13.4	43.4	12.3
Cu-Roug Co. 4.	9.10	68.5	2.82	4.33	2.02	3.65	2.34	2.13	1.83	3.89	8.31	37.2	31.5
Cu-Roug Co. 6.	7.00	77.8	4.16	5.23	4.27	5.38	1.55	5.36	1.02	2.73	3.19	44.9	22.2
Cu-Cl.1 Fd. 7.	8.30	79.5	3.59	5.78	4.73	4.43	3.18	3.06	4.52	4.21	11.7	34.2	20.5
Cu-Cl.1 Co. 12.	13.4	66.8	1.49	2.10	2.32	2.28	2.44	3.56	2.77	3.22	6.69	39.9	33.2
Cu-Cl.2 Co. 14.	18.0	43.6	0.71	1.39	1.64	2.61	1.78	3.17	3.02	3.19	3.65	22.5	56.4
Zn-Roug Fd. 15.	0.26	94.9	58.3	19.8	12.0	0.30	----	1.53	2.87	----	----	0.12	5.05
Zn-Roug Co. 16.	1.14	87.9	21.5	16.7	9.97	10.8	3.48	10.4	4.91	2.87	0.72	6.54	12.1
Zn-Cl.1 Co. 18.	1.18	91.1	19.2	15.8	12.9	15.4	6.89	4.81	5.44	0.28	2.16	8.10	8.91
Zn-Cl.2 Co. 20.	1.06	78.0	19.2	8.66	11.4	10.4	5.60	9.81	4.60	2.16	2.53	3.72	22.0
Cu-Roug Tl. 3.	0.28	94.9	48.2	11.4	3.39	9.76	1.47	7.08	3.30	1.15	6.66	0.20	7.36
Cu-Scav Tl. 5.	0.23	94.7	33.4	17.6	10.8	18.1	11.6	0.41	0.57	0.85	----	1.44	5.29
Cu-Cl.1 Tl. 9.	2.10	92.6	16.1	21.9	15.4	12.1	9.19	3.96	3.29	3.75	4.02	2.83	7.37
Cu-Cl.2 Tl. 11.	3.80	91.9	9.89	16.8	8.82	15.0	10.8	5.84	4.95	1.11	5.04	13.7	8.07
Zn-Cl.1 Tl. 19.	1.14	93.2	27.7	29.7	5.42	6.21	0.57	0.35	5.08	4.56	7.07	6.52	6.82
Zn-Cl.2 Tl. 21.	1.35	80.0	22.7	9.64	15.6	2.32	5.66	2.07	0.38	----	0.90	20.7	20.0
Zn-Roug Tl. 17.	0.22	96.7	45.3	25.7	9.56	14.5	1.10	----	----	----	----	0.52	3.29

TABLE 5

RAW LIBERATION AND CLASSIFICATION DATA FOR SPHALERITE IN THE 20 HBMS MILL PRODUCT SAMPLES.

PRODUCT.	Zn ASSAY.	% LOCKED (0-99.9)	CLASSIFICATION OF UNLIBERATED GRAINS.										% FREE (99.9-100)
			0-10	10-20	20-30	30-40	40-50	50-60	60-70	70-80	80-90	90-99	
Feed 1.	1.91	84.1	7.00	6.79	10.7	4.04	2.06	5.22	1.29	10.3	6.39	30.3	15.9
Cu-Scav Co. 8.	5.30	30.2	0.92	1.94	4.32	0.21	1.59	2.65	0.64	0.66	5.65	11.6	69.8
Cu-Scav Co. 10.	4.80	44.0	7.84	3.18	2.12	2.99	1.41	1.06	0.38	4.22	11.7	9.09	56.0
Cu-Roug Co. 2.	5.80	80.7	6.37	3.58	0.70	3.36	6.46	1.51	11.6	8.37	7.20	31.5	19.3
Cu-Roug Co. 4.	5.60	57.1	3.37	2.99	1.19	4.20	1.87	8.37	3.37	2.50	4.26	25.0	42.9
Cu-Roug Co. 6.	6.00	43.4	5.24	3.57	7.60	5.21	3.68	3.56	7.88	6.78	7.84	12.0	36.6
Cu-Cl.1 Fd. 7.	5.60	69.0	5.41	3.37	0.19	5.16	3.71	1.62	6.65	2.22	3.93	36.8	31.0
Cu-Cl.1 Co. 12.	7.50	56.4	4.75	2.18	2.34	7.99	5.53	6.61	2.40	2.46	4.77	17.4	43.6
Cu-Cl.2 Co. 14.	7.40	39.8	2.40	2.34	1.79	3.07	6.23	1.01	6.15	1.91	3.83	11.1	60.2
Zn-Roug Fd. 15.	1.85	68.3	51.0	0.29	----	----	----	----	15.8	----	1.26	----	31.7
Zn-Roug Co. 16.	30.0	64.8	0.80	1.03	1.77	2.89	2.00	2.23	2.49	3.84	7.27	40.4	35.2
Zn-Cl.1 Co. 18.	41.0	58.0	0.47	1.52	1.99	1.95	1.89	3.00	3.60	4.19	7.13	32.3	42.0
Zn-Cl.2 Co. 20.	46.0	46.3	0.32	0.97	1.29	1.88	2.24	1.97	2.81	2.72	6.50	25.6	53.7
Cu-Roug Tl. 3.	1.62	83.7	11.5	11.2	1.75	8.79	1.01	12.2	4.67	6.08	6.72	19.8	16.3
Cu-Scav Tl. 5.	1.57	67.8	5.83	4.61	3.32	0.20	0.70	20.9	0.86	4.27	0.78	26.3	32.2
Cu-Cl.1 Tl. 9.	4.50	54.1	3.20	2.46	3.20	4.11	2.60	6.57	2.73	8.23	5.43	15.6	45.9
Cu-Cl.2 Tl. 11.	6.60	57.3	4.22	3.33	2.63	2.52	8.47	3.62	2.36	0.60	0.90	28.6	42.7
Zn-Cl.1 Tl. 19.	8.70	75.2	5.51	6.29	10.3	11.8	6.73	11.6	4.20	5.17	5.69	7.85	24.8
Zn-Cl.2 Tl. 21.	23.0	54.0	2.20	3.62	4.89	4.50	5.20	6.86	3.12	5.42	6.17	12.1	46..0
Zn-Roug Tl. 17.	0.70	94.4	20.9	17.6	4.78	11.2	----	3.56	9.13	14.6	6.04	6.59	5.64

Appendix B

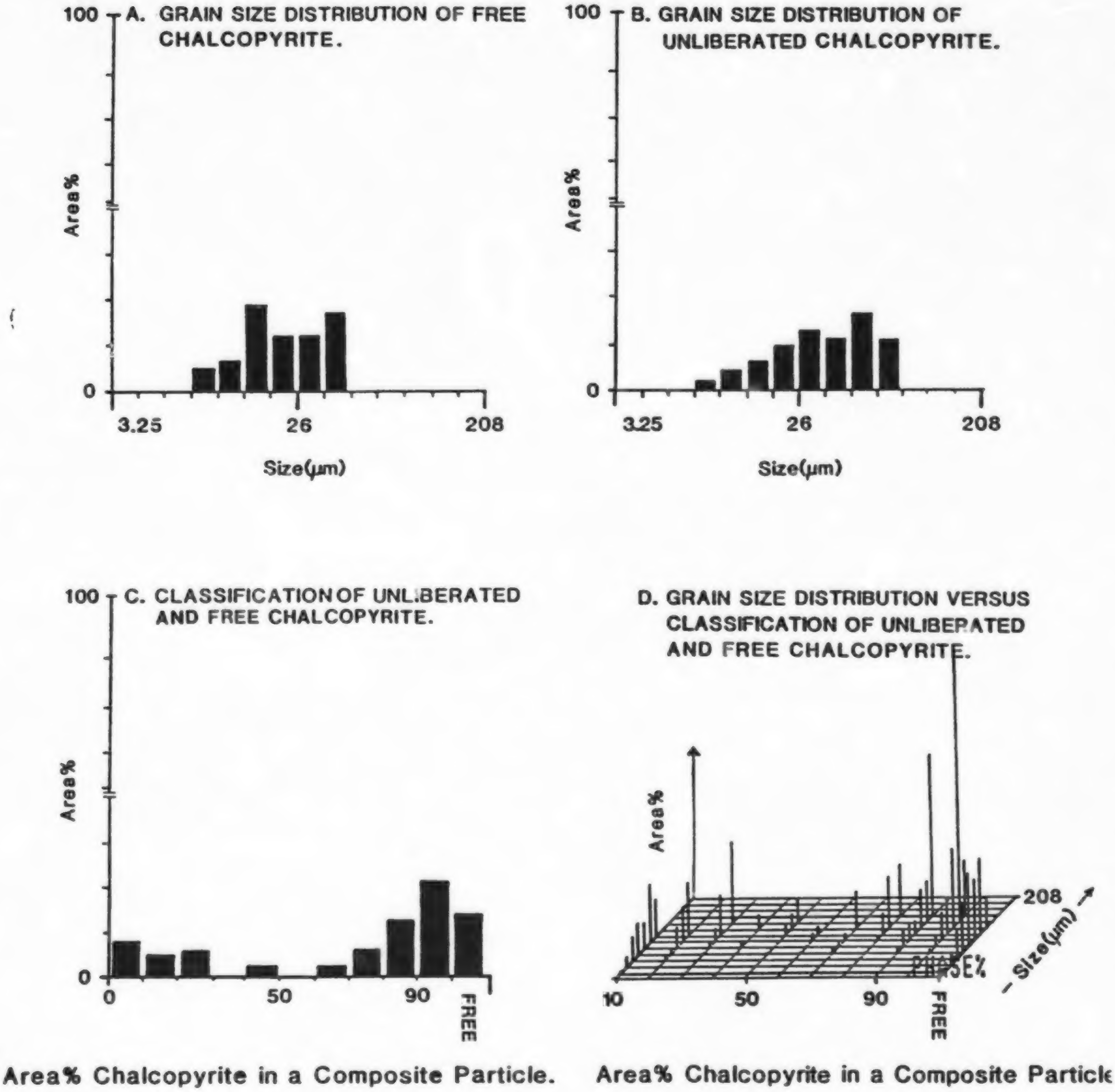

FIGURE 1
PLOT OF THE RAW GRAIN SIZE AND LIBERATION DATA FOR CHALCOPYRITE IN THE HBMS FLOAT FEED.

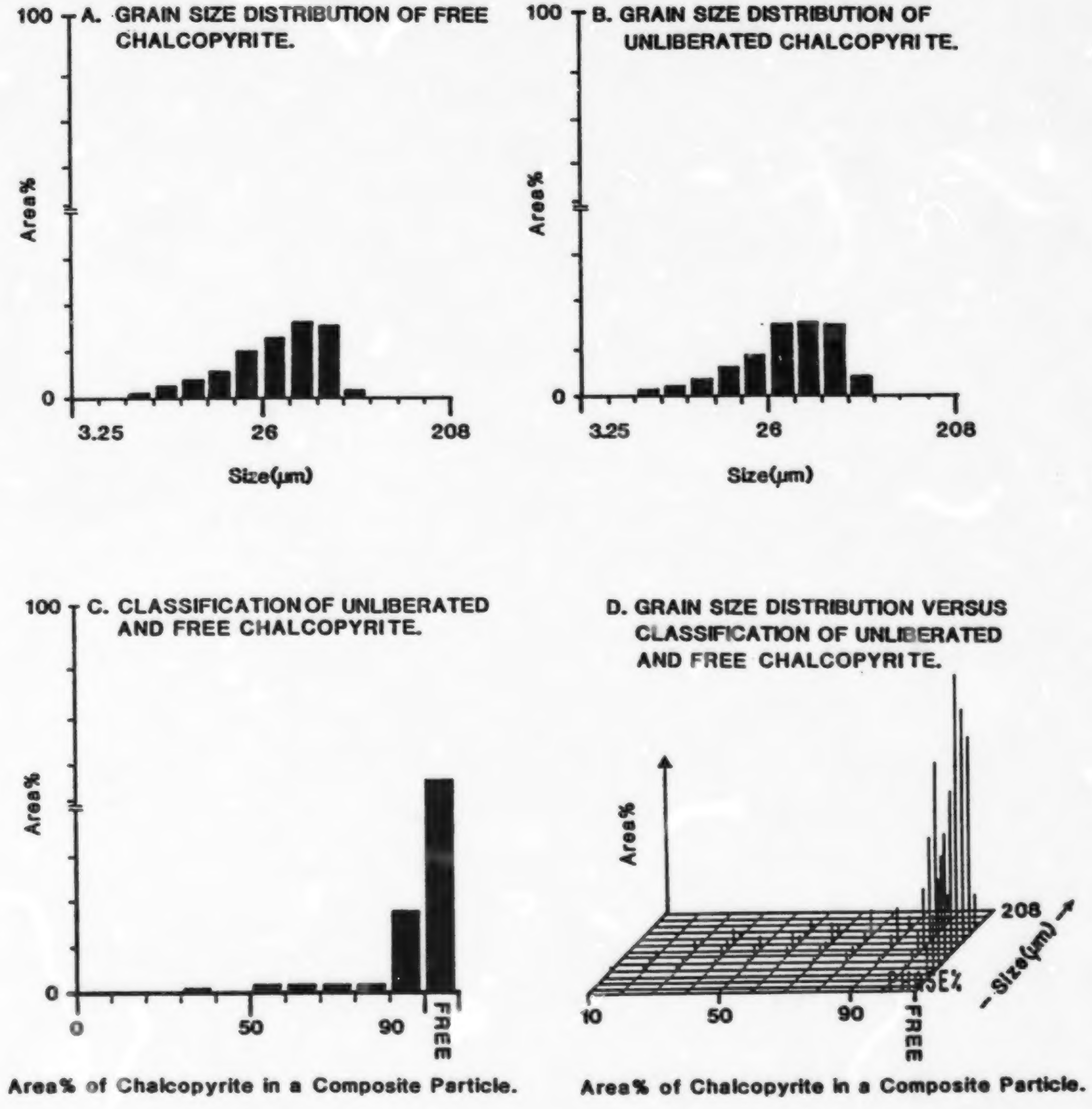

FIGURE 2.
PLOT OF THE RAW GRAIN SIZE AND LIBERATION DATA FOR CHALCOPYRITE IN THE HBMS Cu-CONCENTRATE.

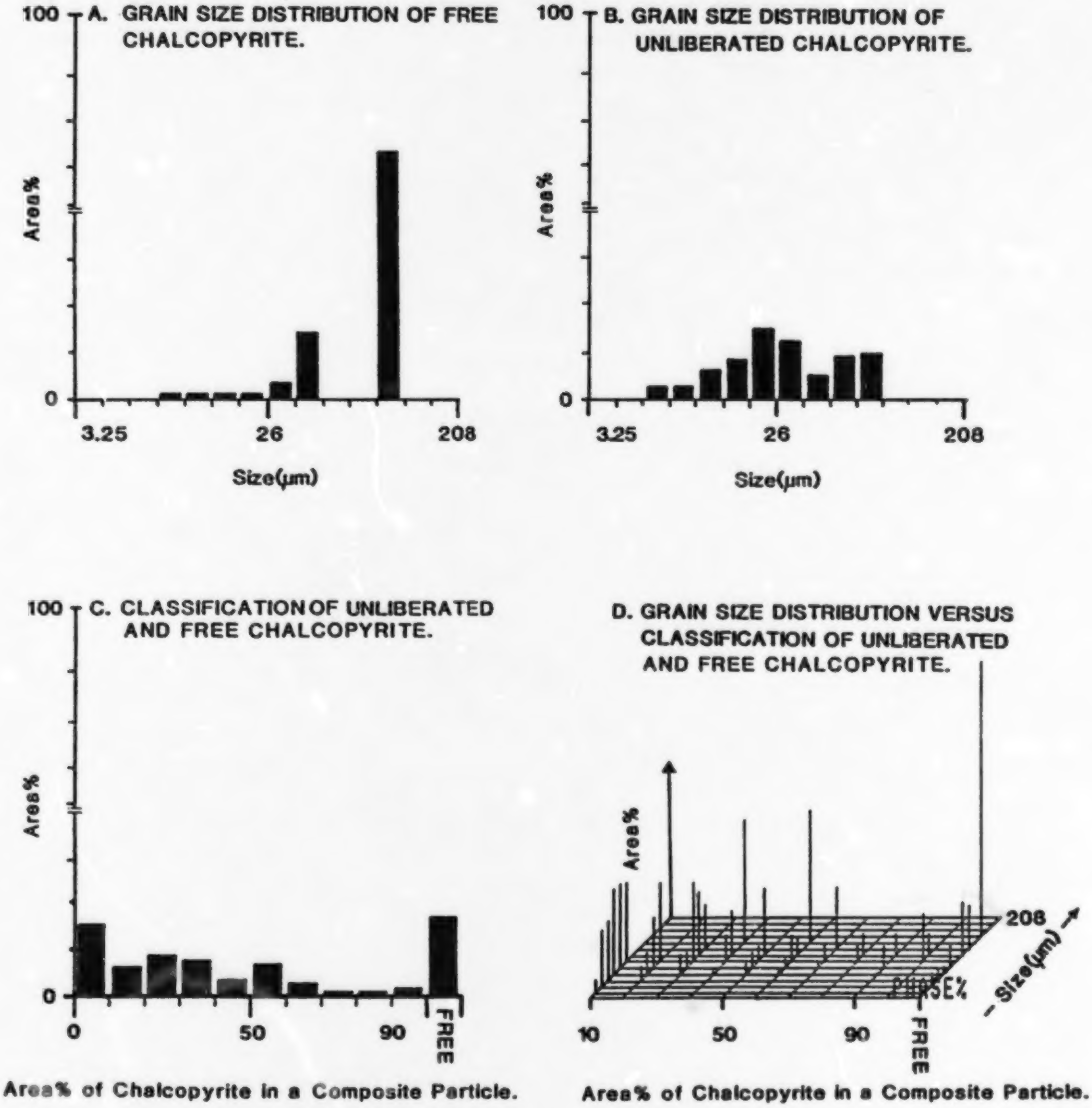

FIGURE 3.
PLOT OF THE RAW GRAIN SIZE AND LIBERATION DATA FOR CHALCOPYRITE IN THE HBMS Zn-CONCENTRATE.

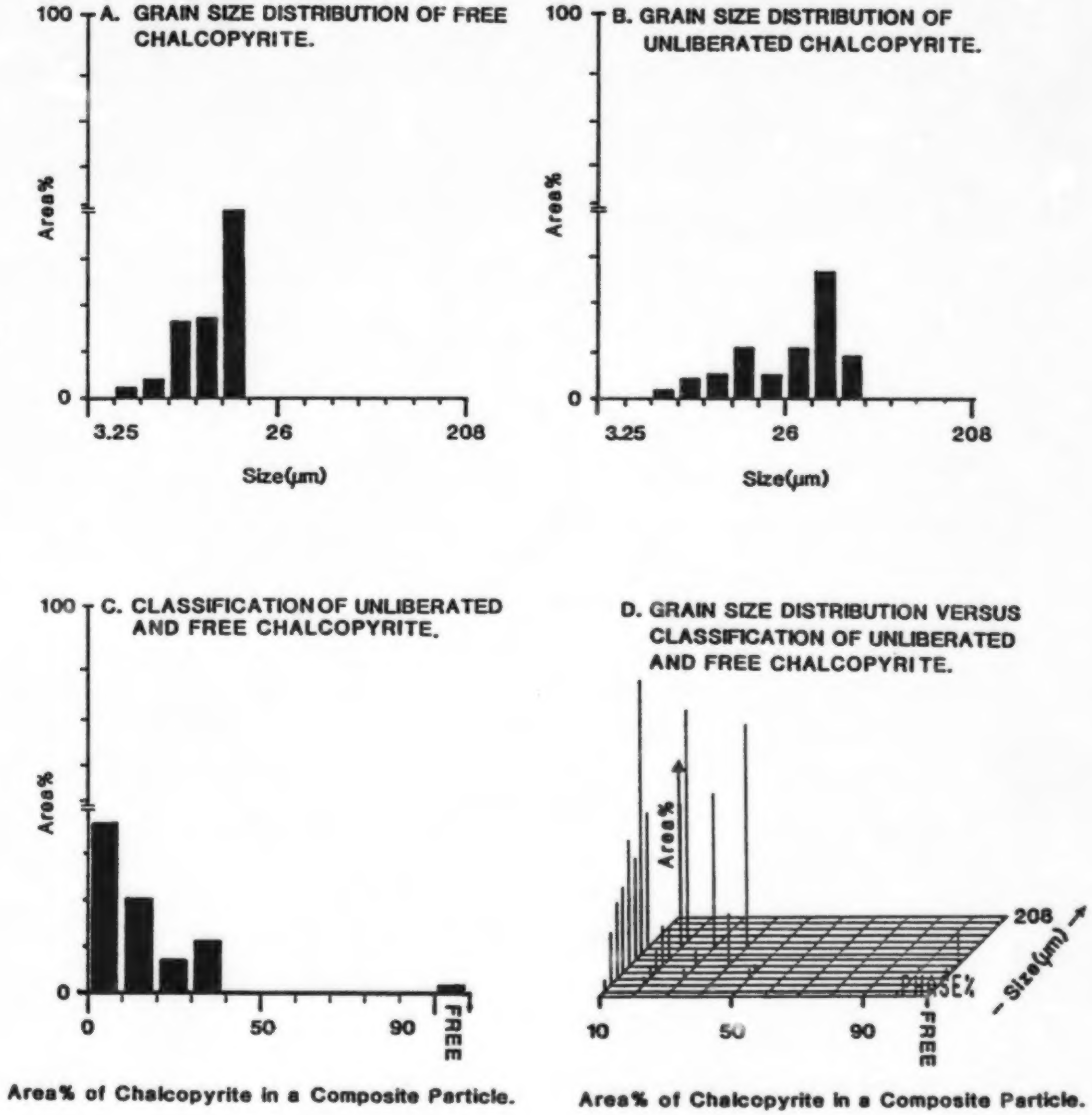

FIGURE 4.
PLOT OF THE RAW GRAIN SIZE AND LIBERATION DATA FOR CHALCOPYRITE IN THE HBMS TAILINGS.

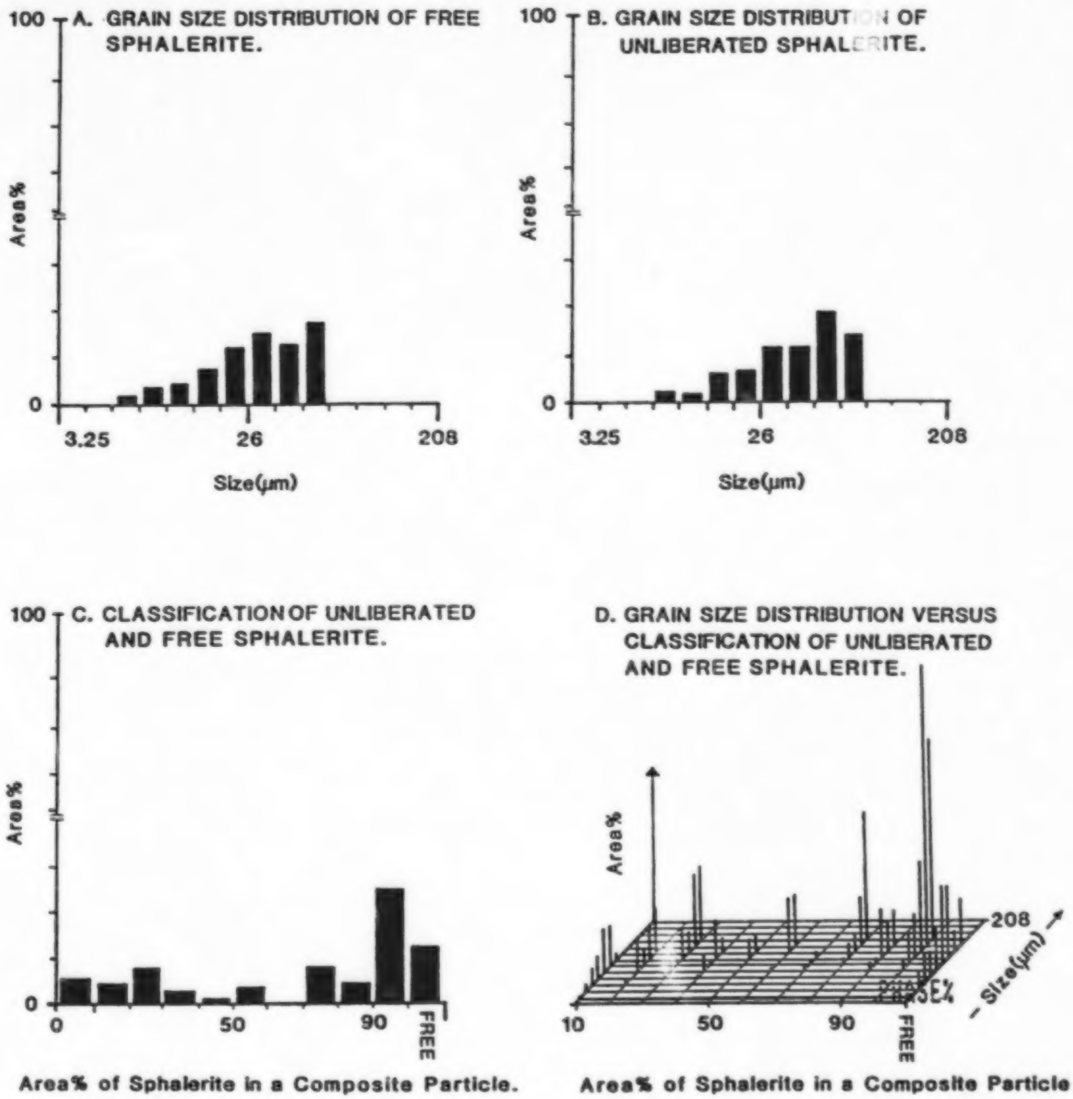

FIGURE 5.
PLOT OF THE RAW GRAIN SIZE AND LIBERATION DATA FOR SPHALERITE IN THE HBMS FLOAT FEED.

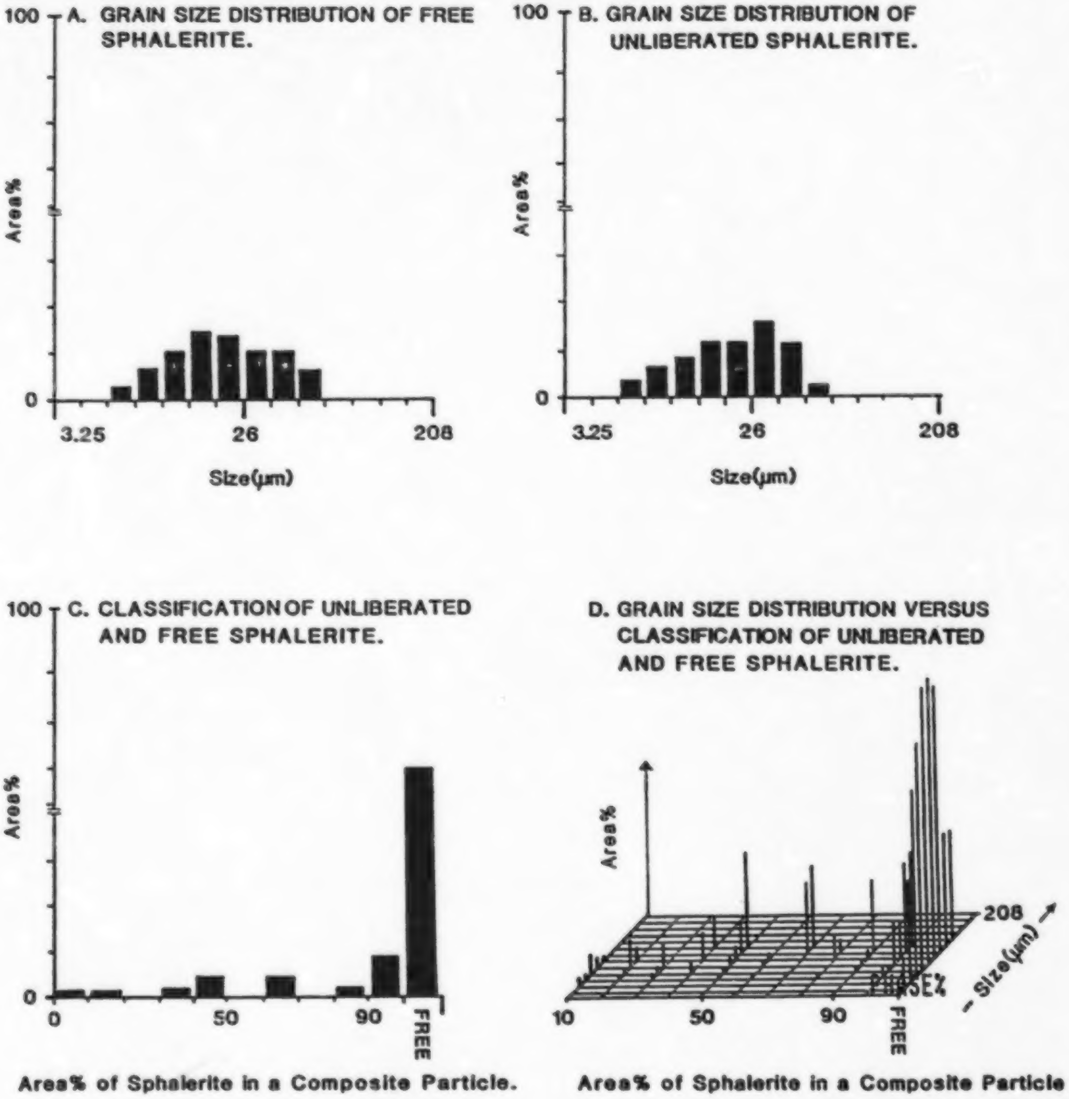

FIGURE 6.
PLOT OF THE RAW GRAIN SIZE AND LIBERATION DATA FOR SPHALERITE IN THE HBMS Cu-CONCENTRATE.

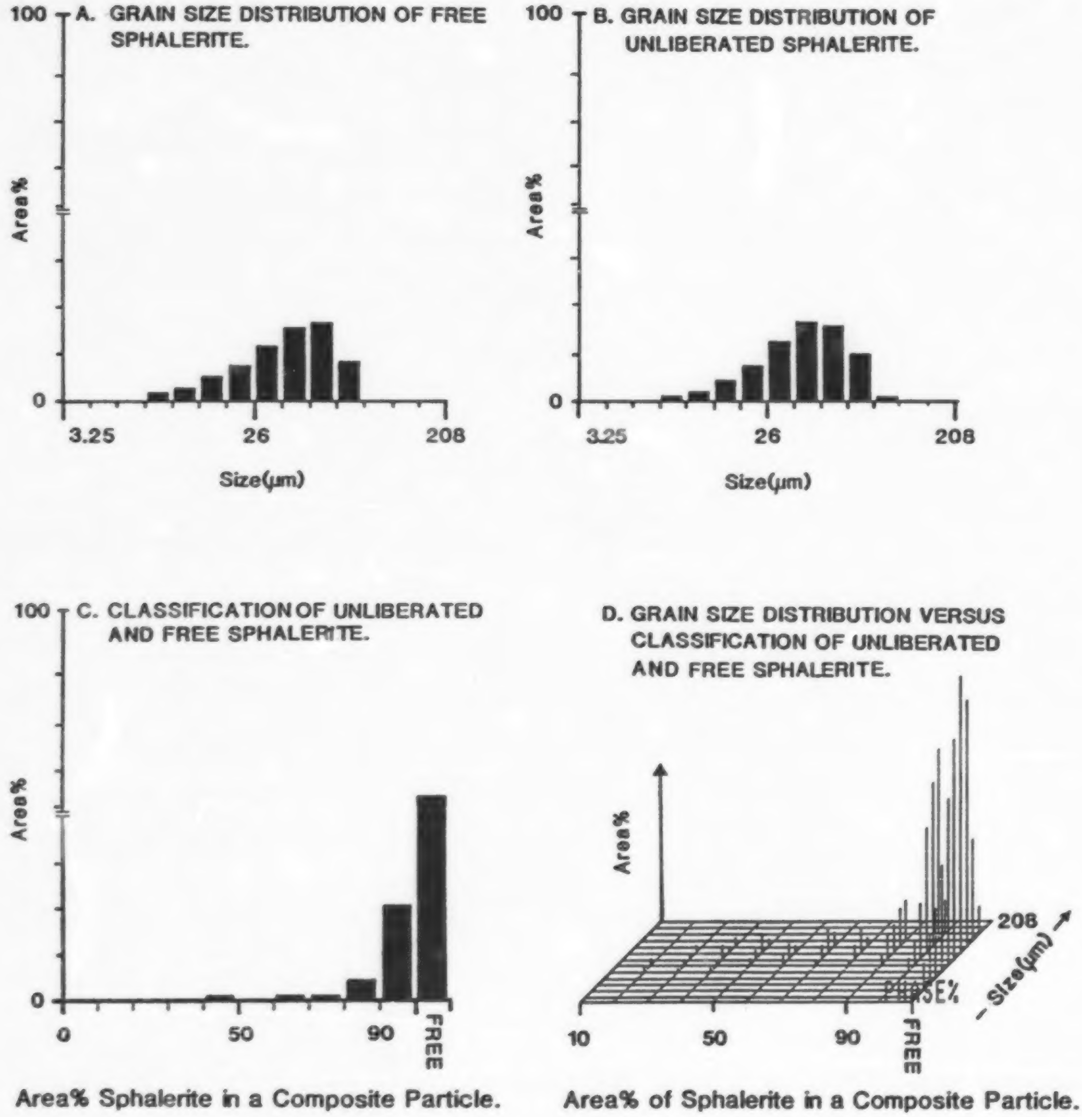

FIGURE 7.
PLOT OF THE RAW GRAIN SIZE AND LIBERATION DATA FOR SPHALERITE IN THE HBMS Zn-CONCENTRATE.

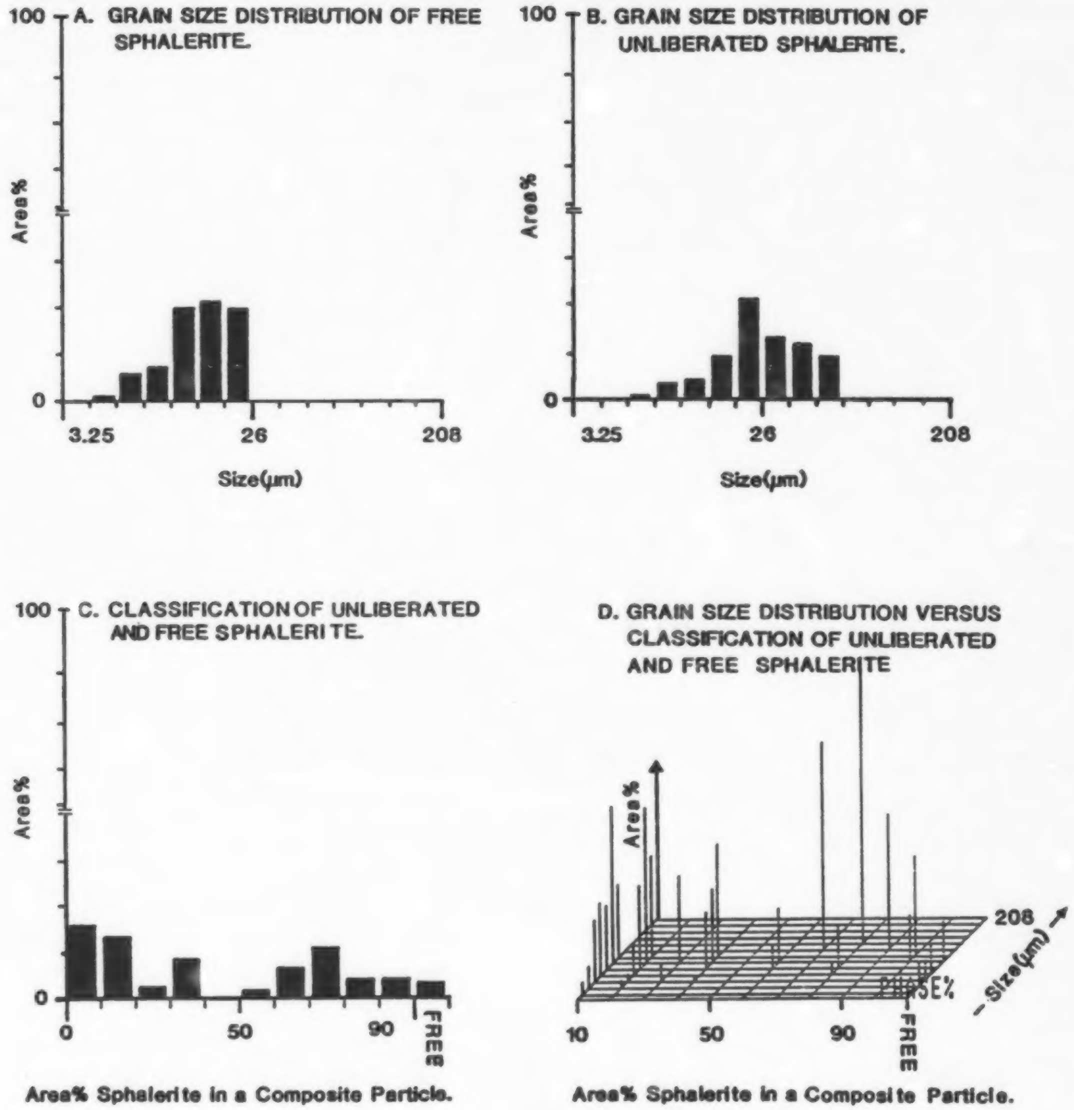

FIGURE 8.
PLOT OF THE RAW GRAIN SIZE AND LIBERATION DATA FOR SPHALERITE IN THE HBMS TAILINGS.

Appendix C

TABLE 1

THE GRAIN SIZE DISTRIBUTION OF ALL PHASES (PARTICLE SIZE DISTRIBUTION) IN THE 20 HBMS MILL PRODUCT SAMPLES.

	Sample.	Classes (um).													Area. (um^2)
		-3.25	-4.60	-6.50	-9.19	-13.0	-18.4	-26.0	-36.8	-52.0	-73.5	-104.	-147.	+147.	
1	Float Feed.	0.04	0.14	0.83	1.33	2.00	2.85	4.23	6.26	10.7	15.1	19.2	18.6	14.7	9620000.
2	Cu Roug Co.	0.04	0.26	1.48	2.18	3.33	4.61	6.94	10.8	15.7	21.1	18.2	10.2	5.00	2640000.
3	Cu Roug Tl.	0.04	0.18	0.80	1.24	1.78	2.48	3.98	6.13	10.5	16.2	20.2	20.2	16.3	6450000.
4	Cu Roug Co.	0.06	0.23	1.84	3.13	4.74	6.75	10.2	13.5	17.1	17.7	13.7	3.61	7.50	2080000.
5	Cu Scav Tl.	0.04	0.16	0.78	1.34	1.81	2.79	4.16	6.32	10.0	14.6	18.8	19.7	19.5	6360000.
6	Cu Roug Co.	0.03	0.17	1.19	2.13	3.21	4.98	7.45	12.4	18.2	18.4	16.4	10.1	5.26	2750000.
7	Cu Cl 1 Fd.	0.04	0.18	1.38	2.14	3.52	5.05	6.61	11.5	16.1	17.4	20.9	9.91	5.30	2740000.
8	Cu Scav Co.	0.11	0.59	3.47	5.93	8.44	12.9	18.1	20.1	17.1	10.1	1.52	1.59	0.00	624000.
9	Cu Cl 1 Tl.	0.05	0.19	1.45	2.51	3.95	5.32	7.46	11.9	17.6	22.0	15.1	10.7	1.80	3320000.
10	Cu Scav Co.	0.08	0.27	1.68	2.93	4.47	6.09	9.90	14.0	15.9	16.0	13.1	9.41	6.26	1820000.
11	Cu Cl 2 Tl.	0.03	0.15	1.16	1.98	2.91	4.39	7.26	10.8	18.5	24.5	19.7	7.14	1.44	2630000.
12	Cu Cl 1 Co.	0.03	0.16	0.90	1.56	2.37	3.93	6.34	11.0	17.5	20.6	18.3	11.8	5.62	2180000.
14	Cu Cl 2 Co.	0.04	0.24	1.56	3.02	4.93	7.57	12.0	16.4	21.5	21.0	8.50	3.24	0.00	1930000.
15	Zn Roug Fd.	0.05	0.25	1.10	1.80	2.59	4.06	5.57	9.68	14.4	19.7	20.6	11.5	8.73	2620000.
16	Zn Roug Co.	0.04	0.28	1.56	2.55	3.84	6.01	9.53	14.2	21.2	21.0	13.6	5.42	0.70	2750000.
17	Zn Roug Tl.	0.04	0.19	0.81	1.33	1.99	3.23	5.22	7.75	12.0	16.3	18.5	19.6	13.0	5550000.
18	Zn Cl 1 Co.	0.02	0.21	1.06	1.79	2.97	4.78	8.07	13.1	18.9	24.9	17.8	6.41	0.00	3220000.
19	Zn Cl 1 Tl.	0.08	0.49	2.58	4.14	6.61	9.25	13.8	16.8	18.3	16.6	8.02	2.45	0.89	2250000.
20	Zn Cl 2 Co.	0.02	0.16	0.87	1.78	2.72	4.60	7.99	13.8	19.5	23.6	16.1	8.41	0.51	3700000.
21	Zn Cl 2 Tl.	0.10	0.56	3.34	5.67	8.59	12.4	16.9	19.7	16.4	11.1	3.20	2.03	0.00	1550000.

TABLE 2

THE GRAIN SIZE DISTRIBUTION OF FREE CHALCOPYRITE IN THE 20 HBMS MILL PRODUCT SAMPLES.

	Sample.	Classes (um).													Area. (um^2)
		-3.25	-4.60	-6.50	-9.19	-13.0	-18.4	-26.0	-36.8	-52.0	-73.5	-104.	-147.	+147.	
1	Float Feed.	0.00	0.11	2.87	7.14	9.69	24.1	16.5	16.9	22.8	0.00	0.00	0.00	0.00	29800.
2	Cu Roug Co.	0.00	1.05	6.44	9.47	11.2	14.7	10.2	13.0	23.0	10.9	0.00	0.00	0.00	79200.
3	Cu Roug Tl.	0.61	1.14	5.87	0.00	17.0	16.3	0.00	59.0	0.00	0.00	0.00	0.00	0.00	1150.
4	Cu Roug Co.	0.01	0.25	3.46	7.87	9.33	12.8	23.4	14.5	19.8	3.24	5.21	0.00	0.00	116000.
5	Cu Scav Tl.	0.00	2.97	8.91	21.0	23.0	44.2	0.00	0.00	0.00	0.00	0.00	0.00	0.00	885.
6	Cu Roug Co.	0.00	0.37	3.26	5.16	8.08	9.61	15.3	13.0	15.6	8.13	13.9	7.65	0.00	128000.
7	Cu Cl 1 Fd.	0.00	0.63	4.35	6.40	8.40	11.7	13.3	22.8	14.7	13.3	4.46	0.00	0.00	114000.
8	Cu Scav Co.	0.00	1.02	6.63	15.8	11.6	15.5	30.5	12.8	6.07	0.00	0.00	0.00	0.00	22300.
9	Cu Cl 1 Tl.	0.00	0.60	4.53	7.35	13.4	8.74	15.6	32.0	17.7	0.00	0.00	0.00	0.00	13900.
10	Cu Scav Co.	0.00	0.88	7.04	13.6	20.5	15.6	19.6	16.9	5.86	0.00	0.00	0.00	0.00	29800.
11	Cu Cl 2 Tl.	0.00	1.52	8.55	14.5	12.0	12.0	19.1	0.00	32.4	0.00	0.00	0.00	0.00	14759.
12	Cu Cl 1 Co.	0.00	0.15	1.41	3.02	4.03	7.14	9.72	14.3	23.3	22.8	6.41	7.62	0.00	337000.
14	Cu Cl 2 Co.	0.01	0.22	2.10	4.07	6.49	8.13	13.6	17.1	21.6	20.9	3.90	1.84	0.00	520000.
15	Zn Roug Fd.	0.00	5.20	22.8	72.0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	219.
16	Zn Roug Co.	0.00	0.82	5.62	4.29	2.30	7.43	8.59	9.71	32.5	28.7	0.00	0.00	0.00	8170.
17	Zn Roug Tl.	0.00	3.67	5.18	20.1	21.2	49.9	0.00	0.00	0.00	0.00	0.00	0.00	0.00	406.
18	Zn Cl 1 Co.	0.19	1.63	5.61	9.02	9.47	3.95	4.04	0.00	66.1	0.00	0.00	0.00	0.00	7510.
19	Zn Cl 1 Tl.	0.66	2.92	15.0	13.6	10.9	19.0	37.9	0.00	0.00	0.00	0.00	0.00	0.00	2250.
20	Zn Cl 2 Co.	0.00	0.11	1.31	2.80	2.13	2.99	2.36	5.05	18.2	0.00	0.00	65.0	0.00	14100.
21	Zn Cl 2 Tl.	0.00	0.94	12.0	19.6	19.2	8.12	0.00	40.2	0.00	0.00	0.00	0.00	0.00	4550.

TABLE 3

THE GRAIN SIZE DISTRIBUTION OF FREE SPHALERITE IN THE 20 HBMS MILL PRODUCT SAMPLES.

	Sample.	Classes (um).													Area. (um^2)
		-3.25	-4.60	-6.50	-9.19	-13.0	-18.4	-26.0	-36.8	-52.0	-73.5	-104.	-147.	+147.	
1	Float Feed.	0.06	0.42	3.34	5.71	6.46	10.6	15.2	19.6	16.3	22.2	0.00	0.00	0.00	53200.
2	Cu Roug Co.	0.00	2.65	8.69	13.3	18.8	21.3	20.8	8.64	5.81	0.00	0.00	0.00	0.00	22500.
3	Cu Roug Tl.	0.00	0.78	5.24	8.25	11.8	20.6	16.3	31.4	5.51	0.00	0.00	0.00	0.00	20300.
4	Cu Roug Co.	0.02	0.29	5.52	11.3	17.6	17.1	17.5	16.8	5.39	8.42	0.00	0.00	0.00	40800.
5	Cu Scav Tl.	0.02	0.25	1.48	5.11	5.68	6.52	8.30	17.6	8.43	14.3	32.3	0.00	0.00	41600.
6	Cu Roug Co.	0.00	1.13	6.59	12.4	12.3	16.5	26.8	13.9	5.98	4.41	0.00	0.00	0.00	48100.
7	Cu Cl 1 Fd.	0.02	0.89	7.08	8.90	15.3	19.4	9.47	15.7	9.96	0.00	13.3	0.00	0.00	46500.
8	Cu Scav Co.	0.00	1.84	8.02	18.3	14.1	24.5	17.2	16.0	0.00	0.00	0.00	0.00	0.00	13500.
9	Cu Cl 1 Tl.	0.01	0.36	2.35	4.84	9.42	10.0	12.7	14.6	16.1	20.3	9.29	0.00	0.00	81200.
10	Cu Scav Co.	0.00	1.29	5.29	16.1	15.2	13.8	21.5	19.2	7.64	0.00	0.00	0.00	0.00	22500.
11	Cu Cl 2 Tl.	0.03	0.55	4.57	7.75	9.40	11.1	18.6	8.23	10.8	15.3	13.7	0.00	0.00	70500.
12	Cu Cl 1 Co.	0.40	2.38	4.97	7.35	8.73	11.1	16.8	18.3	23.4	6.55	0.00	0.00	0.00	74700.
14	Cu Cl 2 Co.	0.02	0.98	4.92	9.52	13.3	18.3	17.1	13.8	13.2	8.85	0.00	0.00	0.00	89000.
15	Zn Roug Fd.	0.10	1.04	3.20	9.82	13.4	14.4	22.5	12.7	22.9	0.00	0.00	0.00	0.00	14500.
16	Zn Roug Co.	0.02	0.63	3.23	5.84	7.43	11.4	15.2	18.6	25.3	11.4	1.02	0.00	0.00	470000.
17	Zn Roug Tl.	0.17	2.95	8.03	10.6	25.1	27.9	25.2	0.00	0.00	0.00	0.00	0.00	0.00	4040.
18	Zn Cl 1 Co.	0.01	0.43	2.07	3.99	5.97	8.22	12.4	16.0	19.9	23.5	7.59	0.00	0.00	840000.
19	Zn Cl 1 Tl.	0.03	1.94	8.01	11.3	12.1	13.0	19.2	14.2	9.83	10.4	0.00	0.00	0.00	73300.
20	Zn Cl 2 Co.	0.01	0.26	1.47	3.12	4.55	7.25	10.4	16.0	20.3	22.0	11.5	1.91	1.23	1530000.
21	Zn Cl 2 Tl.	0.02	1.64	9.00	13.7	15.7	14.3	15.7	13.0	8.45	8.47	0.00	0.00	0.00	207000.

TABLE 4

THE GRAIN SIZE DISTRIBUTION OF UNLIBERATED CHALCOPYRITE IN THE 20 HBMS MILL PRODUCT SAMPLES.

		Classes (um).												
Sample.		-3.25	-4.60	-6.50	-9.19	-13.0	-18.4	-26.0	-36.8	-52.0	-73.5	-104.	-147.	+147.	Area. (um^2)
1	Float Feed.	0.02	0.36	1.20	2.52	4.28	6.25	11.2	14.8	15.6	26.7	17.1	0.00	0.00	160500.
2	Cu Roug Co.	0.01	0.20	0.63	1.18	2.03	4.03	7.21	13.1	21.0	20.6	18.4	4.90	6.77	614000.
3	Cu Roug Tl.	0.01	1.79	4.50	8.71	15.0	16.2	26.5	18.8	8.42	0.00	0.00	0.00	0.00	16210.
4	Cu Roug Co.	0.01	0.15	1.14	2.13	4.20	5.43	10.7	17.3	21.2	28.6	9.17	0.00	0.00	411600.
5	Cu Scav Tl.	0.01	1.48	2.65	5.45	10.7	16.0	18.1	14.4	31.2	0.00	0.00	0.00	0.00	25190.
6	Cu Roug Co.	0.01	0.17	0.65	1.53	2.56	5.36	8.64	10.5	18.6	14.2	12.5	12.7	12.7	479000.
7	Cu Cl 1 Fd.	0.01	0.25	0.74	1.55	2.83	4.53	8.64	11.7	15.6	24.1	18.1	7.41	4.59	497000.
8	Cu Scav Co.	0.00	1.70	5.28	8.35	14.8	19.2	25.7	8.20	17.0	0.00	0.00	0.00	0.00	20400.
9	Cu Cl 1 Tl.	0.03	0.40	1.76	3.46	6.55	12.0	23.0	19.7	20.2	13.0	0.00	0.00	0.00	196400.
10	Cu Scav Co.	0.02	0.50	1.45	3.32	7.01	10.2	16.4	15.5	14.2	13.9	7.64	9.95	0.00	123700.
11	Cu Cl 2 Tl.	0.00	0.28	1.22	2.49	5.47	9.08	14.6	18.0	18.6	21.5	2.57	6.15	0.00	241800.
12	Cu Cl 1 Co.	0.01	0.06	0.34	0.90	1.68	3.46	4.75	9.61	14.7	18.6	26.4	10.7	8.61	729000.
14	Cu Cl 2 Co.	0.01	0.20	0.96	1.90	3.33	6.40	10.0	16.1	22.9	24.2	11.1	3.12	0.00	413000.
15	Zn Roug Fd.	0.26	4.86	8.48	7.00	15.7	30.5	7.48	25.7	0.00	0.00	0.00	0.00	0.00	6344.
16	Zn Roug Co.	0.10	1.75	5.12	7.79	8.31	9.84	12.8	28.8	10.7	4.73	10.0	0.00	0.00	71350.
17	Zn Roug Tl.	0.08	0.81	3.64	6.00	6.88	12.8	7.32	14.7	35.4	12.4	0.00	0.00	0.00	21760.
18	Zn Cl 1 Co.	0.17	1.65	4.36	5.94	7.25	10.2	10.8	15.8	16.9	26.9	0.00	0.00	0.00	86910.
19	Zn Cl 1 Tl.	0.18	2.07	5.60	8.96	9.92	21.4	15.9	25.9	10.1	0.00	0.00	0.00	0.00	42490.
20	Zn Cl 2 Co.	0.06	1.13	4.31	5.42	9.85	13.2	25.1	17.2	7.05	16.5	0.00	0.00	0.00	52200.
21	Zn Cl 2 Tl.	0.01	2.08	6.93	11.2	14.5	13.9	20.8	19.5	11.2	0.00	0.00	0.00	0.00	18880.

TABLE 5

THE GRAIN SIZE DISTRIBUTION OF UNLIBERATED SPHALERITE IN THE 20 HBMS MILL PRODUCT SAMPLES.

	Sample.	Classes (um).													Area. (um²)
		-3.25	-4.60	-6.50	-9.19	-13.0	-18.4	-26.0	-36.8	-52.0	-73.5	-104.	-147.	+147.	
1	Float Feed.	0.02	0.29	0.95	2.05	2.59	8.09	6.78	14.5	16.7	26.5	21.6	0.00	0.00	153100.
2	Cu Roug Co.	0.05	0.55	1.46	2.66	5.53	9.17	15.6	16.4	12.6	13.0	8.02	14.9	0.00	75700.
3	Cu Roug Tl.	0.00	0.49	1.30	2.64	4.06	11.1	12.5	22.6	20.7	24.7	0.00	0.00	0.00	56000.
4	Cu Roug Co.	0.03	0.44	1.84	5.25	7.46	9.70	14.8	16.4	28.4	15.7	0.00	0.00	0.00	48300.
5	Cu Scav Tl.	0.05	0.27	1.24	2.15	4.62	7.79	9.54	18.4	10.1	22.4	23.3	0.00	0.00	56100.
6	Cu Roug Co.	0.03	0.38	1.78	3.38	8.75	11.7	19.2	14.4	25.3	15.1	0.00	0.00	0.00	76800.
7	Cu Cl 1 Fd.	0.00	0.38	1.04	3.41	5.77	8.84	9.65	22.5	20.3	14.3	14.0	0.00	0.00	85600.
8	Cu Scav Co.	0.46	1.66	4.50	15.9	15.6	0.00	37.1	24.7	0.00	0.00	0.00	0.00	0.00	57000.
9	Cu Cl 1 Tl.	0.02	0.28	1.81	4.03	7.21	10.2	9.74	16.5	29.1	21.0	0.00	0.00	0.00	84400.
10	Cu Scav Co.	0.06	1.02	5.64	9.64	11.5	13.2	30.1	16.8	12.1	0.00	0.00	0.00	0.00	14300.
11	Cu Cl 2 Tl.	0.00	0.33	1.43	2.44	4.19	6.72	14.7	20.5	15.0	20.1	14.8	0.00	0.00	88900.
12	Cu Cl 1 Co.	0.23	0.71	1.58	2.68	5.32	5.46	12.8	27.7	16.1	7.36	20.2	0.00	0.00	87400.
14	Cu Cl 2 Co.	0.05	0.83	2.66	5.98	7.97	13.1	13.6	27.0	29.0	0.00	0.00	0.00	0.00	55600.
15	Zn Roug Fd.	0.03	0.21	1.02	2.62	4.26	7.32	16.0	9.93	17.5	20.6	20.7	0.00	0.00	39520.
16	Zn Roug Co.	0.01	0.17	0.51	1.28	2.24	4.41	8.01	13.8	21.3	25.0	12.8	10.7	0.00	798000.
17	Zn Roug Tl.	0.08	0.71	1.59	4.69	5.85	12.9	26.9	18.4	16.1	12.8	0.00	0.00	0.00	22830.
18	Zn Cl 1 Co.	0.00	0.12	0.40	0.95	2.05	4.14	8.07	14.3	20.0	28.9	19.7	1.51	0.00	1080000.
19	Zn Cl 1 Tl.	0.04	0.94	3.52	7.51	10.2	15.2	23.3	25.8	10.8	2.57	0.00	0.00	0.00	149700.
20	Zn Cl 2 Co.	0.01	0.08	0.37	0.84	1.90	4.56	7.71	15.2	22.5	23.1	20.7	3.15	0.00	1210000.
21	Zn Cl 2 Tl.	0.03	1.20	4.09	8.85	12.2	17.6	19.2	20.0	13.6	3.20	0.00	0.00	0.00	202000.

Appendix D

TABLE 1

GRAIN SIZES OF MIDDLING PLUS SUB-MIDDLING CHALCOPYRITE AND SPHALERITE IN SEVERAL POTENTIAL REGRIND STREAMS, AND THE TAILINGS.

SIZES. (um).	STREAM.							
	Cu-Rougher Conc.		Cu-Cl-1. Tail.		Zn-Cl-1 Tail.		Zn-Rougher Tail.	
	Cpy	Sphal	Cpy	Sphal	Cpy	Sphal	Cpy	Sphal
-3.3	0.04	0.09	0.02	0.04	0.02	0.07	0.08	0.10
-4.6	0.53	0.70	0.40	0.48	2.87	1.74	0.81	1.01
-6.5	2.15	2.63	1.91	2.87	9.50	6.01	3.55	2.23
-9.2	3.97	5.36	3.78	6.38	15.2	12.5	6.03	6.58
-13.	6.69	9.83	7.39	9.36	19.7	15.9	6.48	7.25
-18.	10.9	14.4	12.9	14.4	16.2	20.7	12.9	15.3
-26.	16.9	17.6	23.9	8.36	24.7	18.4	7.37	34.7
-37.	19.7	12.9	19.7	21.5	11.8	16.7	14.8	19.2
-52.	18.6	22.1	18.6	29.7	----	8.08	35.6	13.6
-74.	10.7	14.4	11.4	7.00	----	----	12.5	----
-104.	6.89	----	----	----	----	----	----	----
-147.	2.96	----	----	----	----	----	----	----

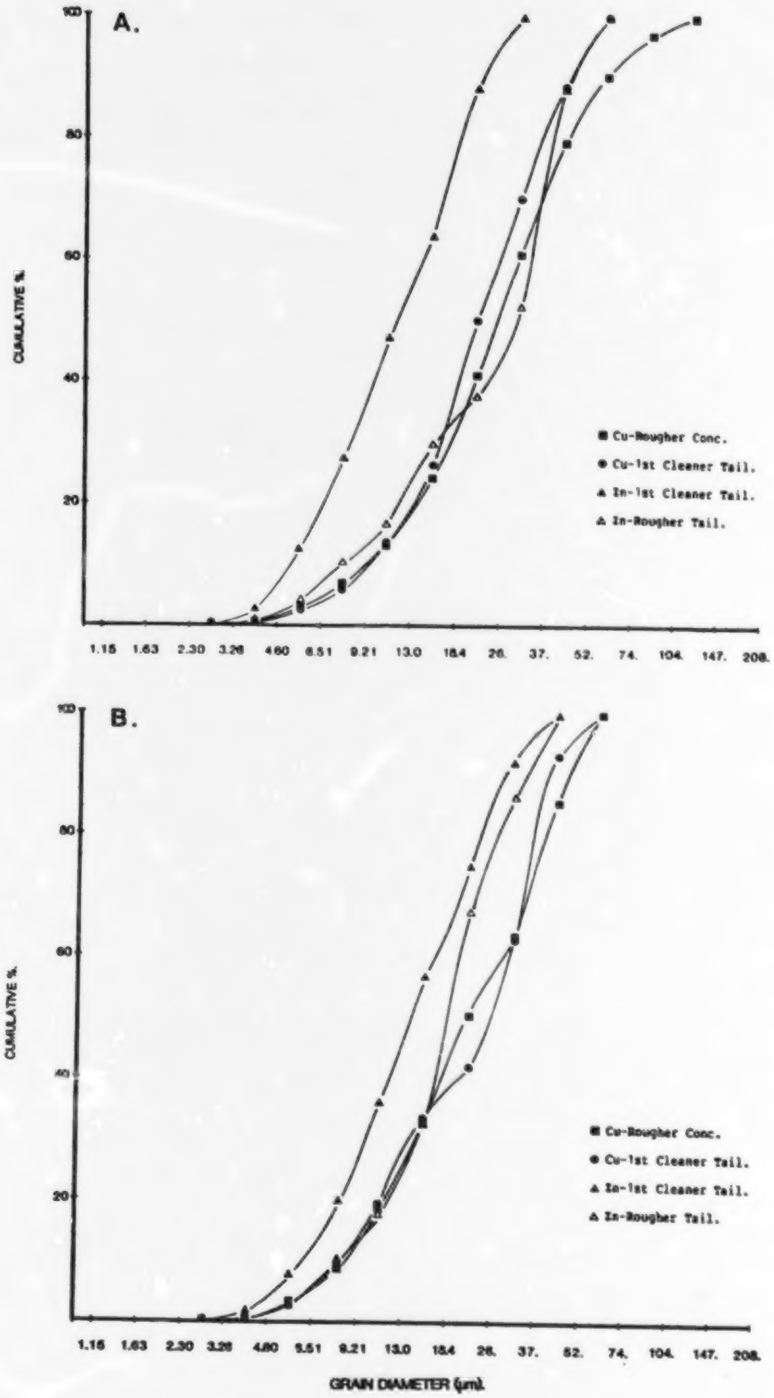

FIGURE 1.
GRAIN SIZE CURVES FOR MIDDLING PLUS SUB-MIDDLING GRAINS OF (A) CHALCOPYRITE AND (B) SPHALERITE, IN SEVERAL POTENTIAL REGRIND STREAMS, AND THE TAILINGS.